U.S. Mines Bureau

[redacted]

Bulletin 173

[redacted]

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

MANGANESE

USES, PREPARATION, MINING COSTS AND THE PRODUCTION OF FERRO-ALLOYS

BY

C. M. WELD and OTHERS

TN
23
U4
no. 173
ENGINEERING
ENGIN STORAGE

LIBRARY
AUG 11 1920
UNIVERSITY OF TORONTO

WASHINGTON
GOVERNMENT PRINTING OFFICE
1920

Bulletin 173

DEPARTMENT OF THE INTERIOR
FRANKLIN K. LANE, SECRETARY
BUREAU OF MINES
VAN. H. MANNING, DIRECTOR

MANGANESE

USES, PREPARATION, MINING COSTS AND THE PRODUCTION OF FERRO-ALLOYS

BY

C. M. WELD and OTHERS

WASHINGTON
GOVERNMENT PRINTING OFFICE
1920

The Bureau of Mines, in carrying out one of the provisions of its organic act—to disseminate information concerning investigations made—prints a limited free edition of each of its publications.

When this edition is exhausted copies may be obtained at cost price only through the Superintendent of Documents, Government Printing Office, Washington, D. C.

The Superintendent of Documents *is not an official of the Bureau of Mines.* His is an entirely separate office and he should be addressed:

SUPERINTENDENT OF DOCUMENTS,
Government Printing Office,
Washington, D. C.

The general law under which publications are distributed prohibits the giving of more than one copy of a publication to one person. The price of this publication is 30 cents.

First edition. March, 1920.

CONTENTS.

Page.

Preface 1

Introduction 3

Chapter 1.—General information regarding manganese, by C. M. Weld 6

  Uses of manganese 6

  Specifications 7

  Prices 9

    Price schedule of May 28, 1918 10

      Silica premiums and penalties 10

      Phosphorus penalty 11

      Payments 11

  Statistics 13

Chapter 2.—Uses of manganese other than in steel making, by W. C. Phalen 15

  Introduction 15

  Uses of manganese dioxide ore 15

    General remarks 15

    The dry cell 15

      General description 15

      Development 16

      Method of manufacture 16

      Constituents used 17

      Functions of different constituents 17

        Rôle of manganese 18

      Character of manganese ore used 18

        Chemical requirements 18

        Physical requirements 19

    Manganese ores in the ceramic industries 19

      Glass making 20

        Chemistry of use of manganese in glass-making process 20

      Specifications for manganese ore used in making glass 21

      Other ceramic uses 21

    Use of manganese salts in driers 22

      Definition 22

      Manganese dioxide 22

      Manganese sulphate 22

      Manganese borate 22

      Manganese resinate 23

      Manganese linoleate 23

      Manganese oxalate 23

    Use of manganese in miscellaneous chemicals 23

      Manganese chloride 23

      Manganese sulphate 24

      Manganese persulphate 24

      Potassium permanganate 24

  Use of manganese in manganese bronze 24

    General remarks 24

    Function of manganese in manganese bronze 25

    Uses of manganese bronze 26

Page.
Chapter 3.—Problems involved in the concentration and utilization of domestic low-grade manganese ore, by Edmund Newton 27
  Introductory statement 27
  Manganese deposits in the United States 27
  Concentration of domestic low-grade manganese ores 30
    Factors controlling the possibilities of concentration 30
      Size and character of deposit 31
      Conditions affecting mining and marketing 31
      Characteristics of ore affecting beneficiation 32
        Character of manganese minerals 32
        Impurities associated with manganese minerals 32
    Concentration processes 36
    Commercial considerations regarding beneficiation 37
      Estimated cost data 38
      Discussion of estimated cost data 43
        Ore 1009 43
        Ore 1011 43
        Ore 1020 43
        Fineness of concentrates 43
Chapter 4.—Preparation of manganese ore, by W. R. Crane 45
  Introductory statement 45
  Methods employed 46
    Dry mining 46
    Washing and concentration 47
  Practice in cleaning manganese ore 48
    Practice at a Virginia plant 48
    Practice at a plant in Georgia 49
    Practice in the Batesville district, Arkansas 50
    Practice at a mill at Philipsburg, Mont 51
    Description of a standard washing plant 51
      Logs 52
      Screens 52
      Picking belts 52
      Jigs 53
      Tables 54
    Dry concentration 54
  General summary of conditions affecting concentration 54
Chapter 5.—Leaching of manganese ores with sulphur dioxide, by C. E. Van Barneveld 57
  Introductory statement 57
  Results of tests 58
  Observations on results of tests 60
    The ore 60
    The pulp 60
    Solubility of the manganese minerals 61
    Iron 61
    Phosphorus 61
    Lime 62
    Zinc and copper 62
    Treatment of the pregnant solutions 62
    Conditions essential in commercial practice 63
    Leaching in two stages 63

Chapters 5.—Leaching of manganese ores with sulphur dioxide—Continued.
  Observations on results of tests—Continued. Page.
    Filters.. 65
    Corrosion.. 65
    Sulphur dioxide.. 65
    Complex ores.. 66
  Costs.. 67
    General estimate.. 67
    Calculated estimates for typical ores.. 68
      Example 1.. 68
      Example 2.. 68
      Example 3.. 69
      Example 4.. 69
  Possible future application of SO_2 method.. 70
Chapter 6.—The Jones process for concentrating magnanse ores; results of laboratory investigations by PETER CHRISTIANSON and W. H. HUNTER.. 71
  Introductory statement.. 71
  Low-temperature reduction.. 72
    Preliminary tests.. 72
      Outline.. 72
      Apparatus used.. 72
      Procedure.. 72
        Charging and heating.. 72
        Discharging.. 72
        Separation.. 73
        Analyses.. 73
      Calculations.. 73
      Conclusions.. 84
      Tabulated results.. 84
      Discussion.. 86
      Products.. 86
      Ores.. 86
      Time.. 86
      Reducing agent.. 86
      Lime.. 87
  Final tests.. 87
    Outline.. 87
  High-temperature reduction tests.. 87
    General outline.. 87
      Charge.. 87
      Pyrometer.. 87
      Crucibles.. 88
      Temperature.. 88
    Tabulated results.. 88
    Discussion.. 89
      General.. 89
      Alloy.. 89
      Recovery.. 89
      Slag.. 90
  Summary.. 90
  Conclusions.. 91

Page.
Chapter 7.—Cost of producing ferro-grade manganese ores, by C. M. Weld and W. R. Crane 92
  Introductory statement 92
  Costs of domestic manganese ores 93
    Carbonate ores 94
    Oxide ores 94
      Cost at the mine 94
        Labor 95
        Supplies 95
        Royalty 95
        Mining and treatment 96
      Cost of transportation to railroad 98
      Summary of cost of oxide ores on board cars 99
    Cost of railroad transportation to market 99
    Summary of cost of domestic ores 101
    Discussion of future possibilities 101
  Cost of foreign ores 102
    Cuba 103
    Brazil 104
    Ocean freight rates 105
    Summary of cost of foreign ores 105
    Discussion of future possibilities 105
  Comparison of domestic and foreign costs 106
Chapter 8.—Production of manganese alloys in the blast furnace, by P. H. Royster 108
  Introductory statement 108
  Furnaces investigated 109
  Collection of data 109
    Character 109
    Completeness 109
    Accuracy 110
  Tabulated operating data 110
  Discussion of data on ferromanganese practice 118
    General description of ferromanganese production 118
      High blast temperature 118
      Basic slag 118
      High hearth temperature 119
      Résumé of conditions in ferromangenese furnace 120
    Theory as to fuel requirements for producing ferromanganese 120
      Composition of furnace gas 122
      "Direct" and "indirect" reduction 123
    Burdening and driving the furnace 124
      Fuel requirements 125
      Percentage of manganese in slag 126
      Rate of driving 127
      Equation for percentage of manganese in slag 128
      Burdening the furnace 131
    Stack loss 132
    Some typical calculations 133
      Cost figures for average practice 134
      Coke ash 134
      Selection of stone 137
    Summary of observations on ferromanganese data 138

Chapter 8.—Production of manganese alloys in the blast furnace—Continued. Page.
- Discussion of data on spiegeleisen practice 139
  - General description on spiegleisen production 139
    - Slag composition 141
    - Hearth temperature 141
    - Composition of furnace gas 142
    - Carbon burned at the tuyères 142
  - Fuel requirements 143
    - Percentage of manganese in the slag 144
    - Stack loss 146
    - Summary of observations on spiegeleisen practice 147

Chapter 9.—National importance of allocating low-ash coke to manganese alloy furnaces, by P. H. Royster 148
- Introductory statement 148
- Furnace data 148
- Importance of allocating low-ash coke to manganese furnaces 148
- Conclusion 150

Chapter 10.—Electric smelting of domestic manganese ores, by H. W. Gillett and C. E. Williams 151
- Introductory statement 151
- Present development of electric smelting of domestic ores 151
- Electric smelting practice on ferromanganese 152
  - Furnaces used 152
  - Raw materials employed 153
  - Average results obtained 153
- Effect of high silica in ores 154
  - Concentration of ores 154
  - Reduction of silicon 154
  - Production of silicomanganese 155
  - Future of high-silica ores dependent on use of silicomanganese 156
- Carbon content of manganese alloys 156
- Reasons for experimental work 157
- Classes of low-grade ores 157
- Experimental work 158
  - Description of furnace used 159
  - Preliminary tests 159
  - Standardization of furnace 159
  - Fluxing tests with high-silica ore 160
  - Advantages of acid slag 162
  - Test of reducers 163
  - Comments on use of rhodochrosite 164
  - Tests of effect of size of reducer constituents 165
  - Slag-melting tests 166
  - Effect of phosphorus content of ores 168
  - Tests with high-phosphorus ores 169
  - Tests on volatilization of phosphorus 170
  - Dephosphorization and smelting in two stages 172
    - Conclusions as to two-stage process 177
  - Tests with high-silica ores 178
    - Applicability of two-stage process to cuyuna ores 179
    - Results of tests 179
- Compilation of data 182
- Conclusion as to electric smelting 183

Page.
Chapter 11.—Use of manganese alloys in open-hearth steel practice, by SAMUEL L. HOYT 187
  Introductory statement 187
  Purpose and scope of investigation 187
  The functions of manganese 189
  Recommendations for the utilization of domestic alloys 191
  "Molten spiegel mixture" practice 191
  High residual manganese practice 192
  Standard open-hearth practice compared with certain other practices 194
  Use of manganese-silicon alloys 195
    Acid practice 195
    Electric-furnace practice 198
    Basic open-hearth practice 198
Selected bibliography on manganese deposits 200
  General 200
  United States 200
  Panama 200
  South America 200
    Brazil 200
    Chile 200
  Europe 200
    Austria 200
    Germany 200
    Russia 201
    Spain 201
  Africa 201
    West Africa 201
  Asia 201
    India 201
    Japan 201
Publications on metallurgy 202
  Publications available for free distribution 202
  Publications that may be obtained only through the Superintendent of Documents 204

TABLES.

Page.
TABLE 1. Range of composition of manganese alloys 6
2. Manganese dioxide and available oxygen in the common manganese minerals 8
3. Carnegie Steel Company's price schedules for domestic ores 9
4. Freight rates on manganese ore from Western States to Chicago 11
5. Comparison of prices of alloy and ore 12
6. Imports of manganese ores, 1903–1918 13
7. Domestic production of manganese ores 13
8. Imports and production of manganese alloys, 1905–1918 14
9. Estimated cost data applied to three California ores 39
10. Results of tests of manganese ores by sulphur dioxide leaching with continuous-drum process 59

Page.

TABLE 11. Results of preliminary low-temperature reduction tests with the Jones process 74
12. Results of final tests with low-temperature reduction 85
13. Results of high-temperature reduction tests 88
14. Wages paid in several of the manganese-producing States 95
15. Cost of handling and hauling ore in manganese-producing States 99
16. Dimensions of 14 of the 18 blast furnaces investigated 109
17. General data from operation of ferromanganese blast furnaces 112
18. General data from operation of spiegeleisen blast furnaces 116
19. Selected data on operation of ferromanganese blast furnaces 121
20. Results of furnace runs arranged to show variation of carbon fuel with weight of slag and with blast temperature 125
21. Results of furnace runs arranged to show variation of percentage of manganese in slag with excess or deficiency of carbon fuel as required for average hearth-temperature conditions 129
22. Results of furnace runs arranged to show variation of percentage of manganese in slag with basicity of slag 129
23. Results of furnace runs arranged to show variation of percentage of manganese in slag with rate of driving 129
24. Results of furnace runs arranged to show manganese lost in stack as related to other factors 133
25. Selected data on operation of spiegeleisen blast furnaces 143
26. Average values for carbon fuel, blast temperature, and slag weight in the furnace runs represented in Table 18 143
27. Results of furnace runs arranged to show variation of manganese in slag according to basicity of slag and according to excess or deficiency of carbon fuel 145
28. Results of furnace runs arranged to show variation of manganese in slag according to rate of driving 145
29. Results of furnace runs arranged to show comparative effects of large and of small stack losses 146
30. Results of slag-smelting tests in electric furnace 167
31. Data showing probable results of electric smelting of domestic manganese ores 184
32. Data showing calculated costs of electric smelting of domestic ores. 186
33. Comparative results obtained with silicomanganese and with a mixture of ferromanganese and ferrosilicon 197

ILLUSTRATIONS.

Page.

FIGURE 1. Flow sheet of proposed plant for SO_2 leaching of manganese ores ... 64
2. Curves showing recoveries of iron and of manganese in heats 16, 17, 32, and 35, with varying temperature 78
3. Curves showing recoveries of iron and of manganese in heat 34 at 1,250° C. and with varying quantities of reducing agent 79
4. Curves showing recoveries of iron and of manganese in heat 17 at 1,150° C. and with varying time 80
5. Curves showing recoveries of iron and of manganese in heat 35 at 1,250° C. and with varying time 81

Page.

FIGURE 6. Curves showing percentages of iron and of manganese in magnetic part in heat 34 at 1,250° C. and with varying reducing agent.... 82
7. Curves showing percentages of iron and of manganese in magnetic part in heats 16, 17, 31, 32, and 35, with varying temperatures... 82
8. Curves showing percentages of iron and of manganese in magnetic part in heat 35 at 1,250° C........ 83
9. Curves showing percentages of iron and of manganese in magnetic part in heat 17 at 1,150° C........ 83
10. Curve showing relation of manganese in slag to carbon difference.. 127
11. Curve showing relation of percentage of manganese in slag to ratio $(CaO+MgO):(SiO_2)$........ 128
12. Curve showing relation of percentage of manganese in slag to rate of driving........ 128
13. Curve showing relation of carbon to silicon in ferro-silico-manganese alloys........ 157

PREFACE.

The history of the domestic production of manganese ores and alloys during the war, in common with that of several other materials equally essential for war purposes, is of interest because of its showing how a hitherto latent industry responded quickly to the spur of necessity.

In the past the supplies of manganese ores used in this country have come largely from Russia and India. More recently imports from Brazil began to assume importance, but up to 1914 they were still a relatively minor factor. Also, small amounts came from Cuba and Central America. The domestic output, however, was practically negligible.

The European war soon shut off Russian sources, and receipts from India declined. The domestic output became somewhat larger, but the deficit was principally made up by greatly increased imports from Brazil.

When this country entered the war, the manganese situation became acute. Manganese being essential in the manufacture of steel, the insuring of an adequate supply was imperative. On account of the increasing difficulty of obtaining foreign ores, through uncertain output and shortage of shipping, it was obvious that this country must turn to its own deposits. In July, 1917, the war minerals committee was established, and surveys of the domestic manganese situation were begun. By the end of the year the domestic output had reached hitherto unprecedented figures.

Early in 1918 the need of diverting every available ship to the transportation of troops and military supplies made necessary the reducing to the minimum of all imports of materials that could be produced in this country. The manganese situation was studied most thoroughly by various governmental bodies, in cooperation with the ferro-alloys subcommittee of the American Iron and Steel Institute, and a program was formulated for reducing overseas imports to the minimum figure consistent with the production of an adequate supply of steel. To stimulate domestic production, a higher price schedule was announced by the American Iron and Steel Institute, with the approval of the War Industries Board. At the same time the market for leaner domestic ores was broadened by lowering the standard grade for manganese alloys.

As a result, had the war continued through the year 1918, there would have been produced more than 300,000 tons of domestic ores containing not less than 35 per cent metallic manganese, with larger quantities than ever before of the lower grade ores suitable for the manufacture of spiegeleisen. In addition, alloys of domestic manufacture of the alloys had entirely replaced those formerly imported. In November, 1918, when hostilities ceased, there was nearly a year's supply of ore and alloys on hand. As a result, the buying stopped almost at once and production fell rapidly. Mines were shut down and furnaces were blown out or used for other products.

Various proposed measures for stabilizing the industry were considered, but none seemed to meet the situation. In 1919 the war minerals relief bill was passed, establishing a commission for investigating losses incurred in the production of manganese and certain other minerals and for reimbursement where the investments were due to representations of the Government.

During the war the engineers connected with the war minerals investigations of the Bureau of Mines kept in touch with developments in the industry. Wherever possible advice and help in mining and milling problems were given by correspondence or in person. Metallurgical problems were carefully studied, not only those bearing on the manufacture of the alloys, but those relating to the use of the alloys in the manufacture of steel. The chief end kept in view was to eliminate waste and to widen and popularize the use of the leaner domestic ores.

In order to supplement the field work and make public the results of its investigations, the Bureau of Mines published a series of mimeographed bulletins dealing with those phases of the industry, which were studied in some detail. The object of this report is to present these papers in more permanent form, in the hope that the information may be of present or future value to the industry.

J. E. Spurr,
Executive, War Minerals Investigations.

MANGANESE USES: PREPARATION, MINING COSTS, AND THE PRODUCTION OF FERRO-ALLOYS.

By C. M. Weld and Others.

INTRODUCTION.

During the past two years the Bureau of Mines has issued a series of mimeographed reports giving the results of research work and experiments conducted as part of its war minerals investigations. In this bulletin the reports on manganese are presented, which range in scope from the beneficiation of the ore to the utilization of the metal. The bulletin is in eleven chapters, each comprising a separate report, arranged in the order given below.

1. General information regarding manganese, by C. M. Weld.
2. Uses of manganese other than in steel making, by W. C. Phalen.
3. Problems involved in the concentration and utilization of domestic low-grade manganese ores, by Edmund Newton.
4. Preparation of manganese ore, by W. R. Crane.
5. Leaching of manganese ores with sulphur dioxide, by C. E. Van Barneveld.
6. The Jones process for concentrating manganese ores; results of laboratory investigations, by Peter Christianson and W. H. Hunter.
7. Cost of producing ferromanganese ores, by C. M. Weld and W. R. Crane.
8. Production of manganese alloys in the blast furnace, by P. H. Royster.
9. National importance of allocating low-ash coke to the manganese-alloy furnaces, by P. H. Royster.
10. Electric smelting of domestic manganese ores, by H. W. Gillett and C. E. Williams.
11. Use of manganese alloys in open-hearth steel practice, by Samuel L. Hoyt.

As each chapter was originally prepared as a unit, there is necessarily some duplication and overlapping. With certain exceptions, however, each paper is reproduced in practically its original form.

In chapter 1, covering certain general phases of the industry, such as uses, specifications, prices, and statistics, the subject matter has been somewhat modified in view of changes in conditions.

About 95 per cent of the manganese consumed in the United States goes into the manufacture of steel. The remaining 5 per cent is used in a number of minor industries, the chief of which is the manufacture of dry cells. These minor uses are described in chapter 2.

No discussion of the geology of manganese ores is presented, but a short bibliography on the subject is appended. With few exceptions the manganese deposits of the United States are irregular, pockety, and uncertain. Largely for this reason, mining methods are crude and hardly warrant descriptions.

The concentration of manganese ores, however, is an important problem and involves some details not common to other ores. Chapter 3 discusses the concentration and utilization of low-grade ores, notes the relationship of concentration to metallurgical practices, outlines concentrating processes, and treats of commercial problems.

In general, as regards concentration methods, manganese ores may be divided into two groups, as follows: (1) Ores permitting mechanical separation of the manganese minerals and the gangue, and (2) ores in which the manganese minerals and the gangues are so intimately associated that separation requires some hydrometallurgical or pyrometallurgical process. Heretofore, attention has been confined almost wholly to mechanical separation, chiefly by gravity. In fact, so far as known, all production of concentrates has been by wet gravity methods, although during the war some companies investigated the commercial possibilities of magnetic separation and reported favorable results, the construction of one magnetic concentration mill being started. The usual wet gravity methods are described in chapter 4.

During the war, the Bureau of Mines undertook investigations of methods applicable to ores of group 2. A hydrometallurgical process involving leaching with sulphur dioxide was studied at the mining experiment station of the bureau at Tucson, Ariz. The results of this work, presented in chapter 5, indicate that the process is metallurgically feasible but the cost makes it unattractive.

The results of an investigation of a pyrometallurgical process known as the Jones process are presented in chapter 6. The process, which is still in the experimental stage, was found to be metallurgically sound. The manganese product is not a concentrate but an alloy, made directly from a lean unconcentrated ore. The customary method is to manufacture alloys from high-grade ores or from concentrates of leaner ores.

A discussion of the costs of ferro-grade manganese ores is presented in chapter 7. This discussion is in general terms only, but should be of use in so far as it relates to competitive conditions.

The results of a study of blast-furnace practice on ferromanganese and on spiegeleisen are presented in chapter 8. On account of the difficulty of obtaining complete and reliable records, the work is to be regarded as largely preliminary. Some of the conclusions tentatively advanced may be erroneous because of the nature of the data on which they are based; in the main, however, they are believed to be correct and are put forward in the hope that they may arouse criticism and thereby stimulate discussion and research. The Bureau of Mines proposes to continue this investigation.

The importance of allocating good grades of coke to manganese-alloy furnaces is discussed in chapter 9. The need for such allocations has passed but it is felt that the paper has more than an historic interest. Conservation and economy would both be served if the principles laid down were more effectively observed.

The chapters already cited deal with the manufacture of manganese alloys in the blast furnace with coke fuel. During the war there was considerable development in the electric smelting of such alloys. The Bureau of Mines was actively interested in the applicability of electric smelting to the leaner, more siliceous domestic ores, and conducted experiments at the Ithaca (Cornell University) field station. The results are presented in chapter 10. The general conclusions are that lean ores and manganiferous slags probably can not be smelted at a profit in the electric furnace except in times of high prices.

The purpose of the investigation described in chapter 11, on the use of manganese alloys in open-hearth steel practice, was chiefly to point the way to conservation of resources. The results show that conservation could best be attained by developing the use of those alloys, such as spiegeleisen and silicomanganese, which could be produced from lean domestic ores, thus conserving the high-grade domestic ores and reducing the need for high-grade foreign ores.

In addition to those members of the staff whose names appear on the several papers, acknowledgment is due to F. H. Probert, G. D. Louderback, Theodore Simons, W. S. Palmer, W. R. Eaton, F. B. Foley, C. F. Julihn, and all others who contributed to the success of the work by their active and whole-hearted cooperation.

CHAPTER 1.—GENERAL INFORMATION REGARDING MANGANESE.

By C. M. Weld.

USES OF MANGANESE.

Approximately 95 per cent of the manganese consumed in this country is used in making steel not only to deoxidize and recarburize the molten metal, thereby making possible the production of cleaner and sounder ingots containing the desired amount of carbon, but also to impart certain qualities to the finished product. Small proportions of manganese, 0.5 to 0.8 per cent, make the steel easier to work and stronger in service. At the same time slight proportions of impurities remaining in the steel are taken into combination and rendered less harmful. Relatively small quantities of so-called "manganese" steel are made which contain 11 to 14 per cent or more of manganese and possess special qualities of hardness and strength.

To be used in manufacturing steel, the ore must first be smelted to an alloy, in which the manganese is combined in varying proportions with iron, carbon, and silicon. Formerly the two standard alloys were ferromanganese, containing 78 to 82 per cent metallic manganese, and spiegeleisen, containing 18 to 22 per cent metallic manganese. During the war the composition of these two alloys underwent considerable modification, the standard manganese content being lowered in order to make practicable the use of leaner domestic ores. At the same time two other alloys, silico-manganese and silico-spiegel, assumed more or less prominence. The approximate range of composition of these four alloys is as follows:

Table 1.—*Range of composition of manganese alloys.*[1]

Alloy.	Mn.	Fe.	Si.	C.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
Ferromanganese	50–80	40–8	0. 5–1. 5	5–7
Spiegeleisen	10–35	85–60	About 1. 0	4–5
Silico-manganese	55–70	20–5	About 25. 0	.35
Silico-spiegel	20–50	67–43	4–10	1. 5–3. 5

The phosphorus must be so low that adding the alloy will not cause the phosphorus in the steel to exceed the specified limit, which is usually about 0.05 per cent. Consequently the phosphorus limit in the alloy will vary more or less directly with its manganese and carbon content.

---

[1] Data largely from Newton, Edmund, manganiferous iron ores of the Cayuga district, Minnesota: Univ. of Minnesota Bull. 5, 1918, 126 pp.

Practice in utilizing the alloy varies widely. The alloy may be added to the molten metal either molten, red-hot, or cold, with or without other additions such as ferrosilicon or coal; it may be added in the open hearth, the converter, or the ladle. The average consumption of metallic manganese per ton of steel is approximately 15 pounds.

Minor uses of manganese ore are as follows: As an oxidizing agent in dry-cell batteries, for decoloration of glass, as a drier in certain paints and varnishes, in the preparation of oxygen on a small scale, and in the manufacture of certain disinfectants. It is also used to color glass, pottery, brick, and tile, and in calico printing and dyeing. Of these minor uses the first two named are by far the most important.

SPECIFICATIONS.

Ores suitable for the manufacture of alloys are generally termed "metallurgical" or "furnace" ores. They may have a wide range in composition, according to the particular alloy to be made from them.

Formerly, when the standard grade of ferromanganese contained 78 to 82 per cent metallic manganese, ores of ferro grade were required to contain not less than 40 per cent metallic manganese, dried at 212° F. Prices were based on not more than 8 per cent silica and not more than 0.2 per cent phosphorus, and ores containing more than 12 per cent silica or more than 0.225 per cent phosphorus were subject to rejection. When, in May, 1918, the standard content of ferromanganese was lowered to 70 per cent, the specified minimum manganese content of ferro-grade ores was correspondingly lowered to 35 per cent metallic manganese. At the same time the rejection limits for silica and phosphorus were considerably extended, but much more severe penalties were imposed for silica exceeding 8 per cent. The rejection of ores with more than 0.25 per cent phosphorus was left to the option of the buyer, who at the same time was urged to use the ore, whatever the phosphorus content, if he could use it to advantage. The effect of silica and phosphorus, and their relation to the alloy to be produced are discussed in chapter 3. The purpose of relaxing specifications was to widen the market for the leaner and more siliceous domestic ores. It is probable that former specifications will be revived with return of normal conditions.

Once it was the custom to pay for the iron in manganese ores of ferro grade, but this practice was discontinued several years ago. The permissible high limit for iron is governed by the nature of the alloy to be produced, as practically all the iron goes into the alloy and degrades it by crowding out manganese. In general, the allowable ratio of iron to manganese in the ore would be approximately 1 to 10 for 80 per cent ferro, or 1 to 5 for 70 per cent ferro.

Ores suitable for the manufacture of spiegeleisen may contain a much larger proportion of iron to manganese and are usually termed manganiferous iron ores. No exact specifications for ores of this class have been customary. In general, the silica and phosphorus requirements have been about the same as for ferro-grade ores, the balance consisting of manganese plus iron in varying ratio, plus gangue materials, such as alumina, lime, and magnesia. Ores containing 15 to 40 per cent metallic manganese were formerly classed as manganiferous, and these limits will probably be adopted again in the future. The war-time classification included ores containing 10 to 35 per cent manganese.

Certain manganiferous ores from the weathered parts of silver-lead deposits contain enough silver to warrant their shipment to lead smelters for use in fluxing. Hence these ores are not available to the steel industry.

When the alloy to be produced is silico-manganese or silico-spiegel a much higher content of silica is acceptable than with ores to be used in making ferromanganese or spiegeleisen, but the former alloys can advantageously be made only in the electric furnace. Ores with 30 to 40 per cent of manganese and 20 to 25 per cent of silica can be used to advantage in making silico-manganese. The ratio of silica to manganese may be still higher if the usual slag-making constituents are relatively absent. It is also probable that at least a part of the phosphorus in the ore may be volatilized in the electric furnace, thus raising the permissible limit of this element. Roughly, the same holds true for silico-spiegel, with the substitution of iron for a part of the manganese.

A large class of manganiferous ores comprises those that are essentially iron ores containing small proportions of manganese, generally about 5 per cent but occasionally as much as 10 per cent. These ores are not available for making manganese alloys, but enter into the manufacture of manganiferous pig iron, which in turn contributes its manganese to the steel made from it.

The ores suitable for oxidizers and chemical use are commonly called "battery," "chemical," or "dioxide" ores. As their function is to act as an agent for carrying oxygen, it is essential that they contain a large proportion of manganese dioxide, either as pyrolusite or other minerals that readily liberate oxygen. The following table shows the proportions of MnO_2 and available oxygen in the common manganese minerals.[a]

TABLE 2.—*Manganese dioxide and available oxygen in the common manganese minerals.*

Mineral.	MnO_2.	Available O.
	Per cent.	*Per cent.*
Pyrolusite	100.00	18.39
Psilomelane	42.46 to 77.33	7.81 to 13.06
Manganite	49.44	9.09
Braunite	43.11	7.93
Hausmannite	37.99	6.99
Rhodochrosite	None.	None.

---

[a] Fermor, L. L., Manganese deposit of India: Geol. Survey of India, Memoirs, vol. 37, p. 598.

The two most important industrial uses of manganese ore as an oxidizer are in the manufacture of dry cells and of glass. Specifications generally call for 80 to 90 per cent MnO_2, but it is understood that ores with as little as 70 per cent MnO_2 were accepted during the war. Formerly it was customary to require that the iron content should not exceed 1 per cent, but here also specifications were greatly relaxed, particularly as it has been shown that the presence of several per cent of iron does not greatly affect the efficiency of the battery. Copper, nickel, and cobalt, on the other hand, are probably harmful when present in excess of a few tenths of 1 per cent, though there is some difference of opinion as to this. The point is discussed in the chapter following.

Leaner argillaceous and siliceous ores, with less than 40 per cent metallic manganese, are used for coloring pottery, tiles, and brick. The amounts used annually for this purpose and for paints, dyeing and printing calicoes, and other purposes are unimportant.

PRICES.

Prices paid for imported metallurgical ores have always been subject to individual contracts, based on the chemical and physical characteristics of the particular ores. The terms of these contracts have, of course, never been made public, but their general trend has no doubt been reflected in the price schedules for high-grade domestic ores issued from time to time by the Carnegie Steel Co. The latter are summarized in the following table:

TABLE 3.—*Carnegie Steel Company's price schedules for domestic ores.*[a]

Year.	Prices in cents per unit for percentages of manganese ranging—				Cents per unit of iron.	Silica standard.	Phosphorus standard.
	From 40 to 43.	From 43 to 46.	From 46 to 49.	49 and more.			
						Per cent.	*Per cent.*
1892	27	28–29	29–30	31	10	8	0.10
1897	25	26–27	27	28	6	8	.10
1903	22	23–24	24	25	5	8	.10
1906	27	28–29	29	30	6	8	.10
1910	23	24–25	25	26	5	8	.20
1914	....	24	25	26	....	8	.20
1915	36	40	43	45	....	8	.20
1916–17	46	50	53	55	....	8	.20
	From 38 to 42.	From 42 to 46.	From 46 to 50.	50 and more.			
1918	80	100	110	120	....	8	.20

[a] Data largely from Mineral Resources U. S., various years, U. S. Geological Survey.

These prices are based on long tons for material delivered at the furnace; material dried at 212° F. silicia penalty, 15 cents per ton for each unit over 8 per cent; phosphorus penalty, up to 1910, 1 cent per unit of manganese for each 0.02 per cent phosphorus over 0.10 per cent; thereafter, 2 cents per unit of manganese for each 0.02 per cent phosphorus over 0.20 per cent.

In May, 1918, a new price schedule was adopted by the Ferro Alloys Committee of the American Iron and Steel Institute and was approved by the War Industries Board. It became effective on May 28. It is given below in full as announced, including penalties and terms of payment, although it is no longer in force and its present value is chiefly historical.

PRICE SCHEDULE OF MAY 28, 1918.

Schedule of domestic metallurgical manganese ore prices per unit of metallic manganese per ton of 2,240 pounds for manganese ore produced and shipped from all points in the United States west of South Chicago, Ill. This schedule does not include chemical ores as used for dry batteries, etc.

Following prices are on the basis of delivery f. o. b. cars South Chicago, and are on the basis of all-rail shipments. When shipped to other destination than Chicago, the freight rate per gross ton from shipping point to South Chicago, Ill., is to be deducted to give the price f. o. b. shipping point.

Prices for ore dried at 212° F.

Content of metallic manganese, per cent.	Price per unit	Content of metallic manganese, per cent.	Price per unit.
35 to 35.99	$0.86	45 to 45.99	$1.12
36 to 36.99	.90	46 to 46.99	1.14
37 to 37.99	.94	47 to 47.99	1.16
38 to 38.99	.98	48 to 48.99	1.18
39 to 39.99	1.00	49 to 49.99	1.20
40 to 40.99	1.02	50 to 50.99	1.22
41 to 41.99	1.04	51 to 51.99	1.24
42 to 42.99	1.06	52 to 52.99	1.26
43 to 43.99	1.08	53 to 53.99	1.28
44 to 44.99	1.10	54 and over	1.30

For manganese ore produced in the United States and shipped from points in the United States east of South Chicago, 15 cents per unit of metallic manganese per ton shall be added to above unit prices.

Above prices are based on ore containing: Not more than 8.00 per cent silica; not more than 0.25 per cent phosphorus, and subject to:

SILICA PREMIUMS AND PENALTIES.

For each 1 per cent of silica under 8 per cent down to and including 5 per cent, premium at rate of 50 cents per ton. Below 5 per cent silica, premium at rate of $1 per ton for each 1 per cent.

For each 1 per cent in excess of 8 per cent and up to and including 15 per cent silica there shall be a penalty of 50 cents per ton; 15 per cent and up to and including 20 per cent silica, there shall be a penalty of 75 cents per ton.

For ore containing in excess of 20 per cent silica, a limited tonnage can be used; but for each 1 per cent of silica in excess of 20 per cent, and up to and including 25 per cent silica, there shall be a penalty of $1 per ton.

Ore containing over 25 per cent silica subject to acceptance or refusal at buyer's option, but if accepted shall be paid for at the above schedule with the penalty of $1 per ton for each extra unit of silica.

All premiums and penalties figured to fractions.

PHOSPHORUS PENALTY.

For each 0.01 per cent in excess of 0.25 per cent phosphorus there shall be a penalty against unit price paid for manganese of one-half cent per unit figured to fractions.

In view of existing conditions, and for the purpose of stimulating production of domestic manganese ores, there will be no penalty for phosphorus so long as the ore shipped can be used to advantage by the buyer. The buyer reserves the right to penalize excess phosphorus as above by giving 60 days' notice to the shipper.

The above prices to be net to the producer; any expenses, such as salary or commission to buyer's agent, to be paid to the buyer.

Settlements to be based on analysis of ore sample dried at 212° F. The percentage of moisture in ore samples as taken to be deducted from the weight.

PAYMENTS.

Eighty per cent of the estimated value of ore (less moisture and freight from shipping point) based on actual railroad scale weights to be payable against railroad bill of lading with attached certificates of sampling and analysis of an approved independent sampling chemist, balance on receipt of ore by buyer.

Actual railroad scale weights to govern in final settlement.

Cost of sampling and analysis to be equally divided between buyer and seller.

It will be observed that the value per long ton of natural (undried) manganese ore f. o. b. mine was to be calculated from this schedule in the following manner; (1) Multiply the percentage of manganese in material dried at 212° F. by the corresponding unit price as given in the table, adding 15 cents to this price if the ore was produced east of Chicago; (2) add premiums or deduct penalties, if any, for silica and phosphorus; (3) convert to wet or natural basis; (4) deduct freight per long ton to Chicago.

The following freight rates from points in the several Western manganese-producing States to Chicago were still in force in December, 1919. These were for carload lots with a minimum weight of 60,000 pounds.

TABLE 4.—*Freight rates on manganese ore from Western States to Chicago.* [a]

State.	Rate per ton of 2000 pounds.	Rate per ton of 2240 pounds.
Oregon	$11.00	$12.32
Washington	11.00	12.32
California	11.00	12.32
Montana	8.00	8.96
Arizona	9.00	10.08
Colorado	7.00	7.84
Nevada	10.00	11.20
Utah	9.00	10.08
New Mexico	7.00	7.84

[a] Personal communication from H. H. Porter of U. S. Shipping Board.

The ore prices given above applied only to ferro-grade ores. Prices for manganiferous ores have usually been subject to individual contract. In 1918 the better class of Leadville ores sold for about 30 cents per unit, with a premium of 6 cents per unit for iron and a moderate penalty for silica.

Comparison of the prices for ferro-grade ore with those for ferromanganese is of interest. Eighty per cent ferromanganese was worth $40 to $60 per ton in prewar times. The price from May to November, 1918, was $250 for 70 per cent material. In the tables below are shown (1) the price of ferro per ton; (2) the price per unit of metallic manganese in ferro; (3) the price per unit of 46 per cent manganese ore; (4) the unit price of metallic manganese in ferro divided by the unit price of metallic manganese in the ore, to show the ratio between the two. Since November, 1918, the market has been in a demoralized condition, quotations falling continuously and rapidly.

TABLE 5.—*Comparison of prices of alloy and ore.*[a]

Year. (1)	Price of 80 per cent ferro per ton. (2)	Price per unit of manganese. (3)	Price per unit of domestic ore containing 46 per cent manganese, delivered at furnace. (4)	Column 3 divided by column 4. (5)
1910	$40.49	$0.51	$0.25	2.04
1911	37.25	.44	.25	1.76
1912	50.40	.63	.25	2.52
1913	57.87	.72	.25	2.88
1914	55.80	.70	.25	2.80
1915	92.21	1.15	.40	2.88
1916	164.12	2.05	.50	4.10
1917	300.17	3.86	[b].75	5.15
1918	285.00	3.56	[b]1.23	2.93

[a] Data largely from Metal Statistics, 1918, published by the American Metal Market and Daily Iron and Steel Report.

[b] Estimated average.

As the last column in the Table 5 indicates, although the price of ore relative to alloy fell far below during 1916 and 1917, the price adjustments of 1918 brought this ratio back again nearly to the figures for 1913–1915; a fact that appears to controvert the opinion prevalent among producers that the proportionate prices of 1918 strongly favored the furnaces. In fact, the ratio shown for 1918 was possibly less favorable to the furnaces than that for 1913–1915, owing to the great increase of conversion cost through high costs of labor, fuel, and supplies.

The prices of "battery" ore also rose enormously during the war. Such ores were formerly worth $20 to $35 per ton. The best grades at last accounts were being sold for $80 to $110 per ton delivered.

STATISTICS.

In the following tables of imports, production, etc., the number of tons of contained metallic manganese are shown as well as the number of tons of ore and alloys, for in no other way can a true comparison be made. The metallic manganese in the ores is also multiplied by a factor representing the conversion loss from ore to alloy to give the "recoverable" manganese in a form suitable for use in steel manufacture. The recovery in making ferromanganese is assumed to be 73 per cent, and in making spiegel 65 per cent. Where the exact analysis of the ores and alloys is not known, the figures for tons of contained metallic manganese are estimated on the basis of the best data available.

TABLE 6.—*Imports of manganese ores, 1903–1918.*

(Long tons.)

Year.	Russia.	India.	Brazil.	Other countries.[a]	Total ores.	Total manganese.	Recoverable manganese, 73 per cent.
1903	5,576	35,900	76,910	....	146,056	55,350	40,450
1904	11,959	10,200	66,875	....	108,579	41,200	30,100
1905	24,650	101,030	114,670	....	277,033	114,340	83,500
1906	13,805	154,189	30,260	....	221,260	97,600	71,350
1907	1,000	95,300	52,922	....	209.021	71,950	52,500
1908	250	143,813	17,150	....	178,203	79,760	58,150
1909	14,486	145,140	35,690	....	212,765	95,800	70,000
1910	33,120	140,965	55,750	....	242,348	110,250	80,500
1911	19,103	106,580	41,600	....	176,852	81,590	59,600
1912	83,334	128,645	81,580	....	300,661	142,700	104,200
1913	124,337	141,587	70,200	8,976	345,090	168,142	123,000
1914	52,681	103,583	113,924	13,106	283,294	134,640	98,300
1915	....	36,450	268,786	8,749	313,985	142,680	104,000
1916	....	51,960	471,837	52,524	576,321	259,317	189,300
1917	....	48,975	512,517	68,450	629,942	282,500	206,400
1918	....	29,275	345,877	116,151	491,303	204,137	149,200

[a] Chiefly Cuba and Central America. In 1918 the sources were as follows (figures are long tons): Cuba, 82,974; Costa Rica, 9,680; Panama, 5,607; Mexico, 5,251; United Kingdom (probably reexported Indian), 4,362; Chile, 2,998; Argentine, 849; China, 3,000; Japan, 709; miscellaneous small lots, 721.

TABLE 7.—*Domestic production of manganese ores.*[a]

(Tons of 2,240 pounds.)

Item.	1912	1913	1914	1915	1916	1917	1918
Ferro grade:[b]							
Total ores	1,664	4,048	2,365	9,709	26,997	114,216	204,497
Total manganese	765	1,860	1,090	4,470	11,880	48,000	118,000
Recoverable manganese	560	1,360	795	3,260	8,670	35,000	86,000
Spiegel grade:[c]							
Total ores[d]	107,569	138,089	158,582	207,511	364,486	732,618	831,000
Total manganese	20,994	20,642	25,170	32,093	58,318	116,880	125,000
Recoverable manganese	13,650	13,420	16,360	20,850	37,900	76,000	81,200
Total recoverable manganese, both grades	14,210	14,780	17,155	24,110	46,570	111,000	167,200

[a] Data in part from Hewett, D. F., Our mineral supplies; manganese: U. S. Geol. Survey Bull. 666–C, 1917, p. 4, and in part from Mineral Resources U. S., U. S. Geol. Survey, for several years.

[b] Ferro grade, 1913–1917, 40+ per cent manganese; in 1918, grade was dropped to 35+ per cent manganese.

[c] Spiegel grade, 1913–1917, 15 to 40 per cent manganese; in 1918, grade was dropped to 10 to 35 per cent manganese.

[d] Includes manganiferous zinc residuum, but excludes manganiferous ores.

TABLE 8.—*Imports and production of manganese alloys, 1905–1918.*[a]

[Tons of 2,240 pounds.]

Year.	Ferromanganese.				Spiegeleisen.				Total manganese, both alloys.	Steel produced.
	Imports.	Production.	Total alloy.	Total manganese.	Imports.	Production.	Total alloy.	Total manganese.		
1905......	52,841	62,186	115,027	92,020	55,457	227,797	283,254	56,650	138,670	20,023,947
1906......	84,359	55,520	139,879	112,000	103,268	244,980	348,248	69,650	18,1650	23,398,136
1907......	87,400	55,918	143,318	114,800	48,994	283,430	332,424	66,500	181,300	23,362,594
1908......	44,624	40,642	85,266	68,150	4,579	111,376	115,955	23,200	91,350	14,023,247
1909......	88,934	82,209	171,143	137,000	16,921	142,831	159,752	31,950	168,950	23,955,021
1910......	114,228	71,376	185,604	148,300	25,383	153,055	178,438	35,700	184,000	26,094,919
1911......	80,263	74,602	154,865	123,900	20,970	104,013	124,983	25,000	148,900	23,676,106
1912......	99,137	125,378	224,515	179,700	1,015	102,561	103,576	20,700	200,400	31,251,303
1913......	128,070	119,495	247,565	198,050	77	110,338	110,415	22,000	220,050	31,300,874
1914......	82,997	106,083	189,080	151,250	2,870	79,935	82,805	16,560	167,810	23,513,030
1915......	55,263	149,521	204,784	163,800	200	97,885	98,085	19,600	183,400	32,151,036
1916......	90,928	221,532	312,460	249,970	....	194,002	194,002	38,800	288,770	42,773,680
1917......	45,381	260,125	305,506	244,400	....	192,985	192,985	38,600	283,000	45,060,607
1918......	26,200	333,027	357,227	274,000	....	283,853	283,853	51,000	325,000	42,212,000

[a] Data compiled from various sources, largely from the statistical bulletins of the American Iron and Steel Institute.

The figures in these tables tell briefly the history of the manganese industry for a number of years past. Thus, Table 6 shows how imports from Russia ceased early in the war and imports from India declined, whereas imports from Brazil increased enormously up to 1918, when restrictions on account of the shortage of ships caused a decrease. During 1918, however, imports from Cuba and from Central American countries practically doubled the 1917 figure.

The domestic industry shows the first effects of the war in 1916. From that time on, domestic production, both of ores and of ferromanganese, increased rapidly.

Table 8 shows that dividing the total pounds of metallic manganese in all alloys produced and imported during the 14 years, 1905 to 1918, by the total tons of steel produced during the same period gives a quotient of 15.5 pounds of manganese per ton of steel. As there were heavy stocks on hand at the end of 1918, it is probable that the true average figure for metallic manganese consumed per ton of steel is more nearly 15 pounds.

No comprehensive figures showing the consumption of "chemical" ore are available. The requirements of the United States are understood to be 25,000 to 40,000 tons annually. In the past these ores were largely imported from Russia. Recently they have come in part from domestic sources and in part from Japan, Brazil, and other countries.

CHAPTER 2.—USES OF MANGANESE OTHER THAN IN STEEL MAKING.

By W. C. Phalen.

INTRODUCTION.

This report is divided into two parts—use of manganese dioxide ore; use of manganese bronze.

USES OF MANGANESE DIOXIDE ORE.

GENERAL REMARKS.

Estimates of the quantity of manganese ore used in the industries, other than metallurgical, range from 25,000 to 50,000 tons per annum. The latter figure is probably more nearly correct. Most of the ore used is the highest grade of pyrolusite (MnO_2) obtainable and commands a much higher price than metallurgical ore. The manganese dioxide ore is generally termed chemical manganese ore.

Manganese ore, as an oxidizing agent, is used in the manufacture of dry cells, as a decolorizer in certain kinds of glass, and as a drier in oils, paints, and varnishes. The ore is also used directly or indirectly in the manufacture of various manganese chemicals. Many of the textbooks describe its reaction with hydrochloric acid as a source of chlorine. It is no longer used commercially in making chlorine gas, as the latter is an abundant by-product in the manufacture of caustic soda and potash. Some chlorine may be made by this method when the object is the production, not of chlorine, but of manganese chloride. The ore has been mentioned as a soil stimulant, but its value in the fertilizer industry has not been established, and so far as known it is not a constituent of any commercial fertilizers. Only those uses of commercial importance are mentioned here.

THE DRY CELL.

GENERAL DESCRIPTION.

To show the importance of manganese dioxide in the dry cell, the general make-up of the cell, the chemistry of the processes involved, and the constituents entering into the reactions are described herewith.

The modern dry cell, commonly but erroneously called the dry battery, is a portable modification of the "Disque" Leclanche cell invented by Leclanche about 1868. That consisted of a porous cup containing a carbon electrode and a mixture of rather coarsely ground retort coke and pyrolusite. The cup was sealed, two vents being left for the escape of gases, and was placed, with a zinc rod, in a glass jar containing a solution of ammonium chloride. Because of the simplicity and ease of operation of this cell, many attempts were made to fix the electrolyte in various media, such as sawdust, gelatin, asbestos, and silicic acid, so as to make the cell portable.

In 1888 Gassner brought out the first really successful dry cell. The positive pole consisted of a cylindrical mass of ground pyrolusite and coke packed in a canvas bag around a carbon electrode. This was placed in a zinc container which also served as the negative pole, and a paste of plaster of Paris, zinc chloride, and ammonium chloride was poured under and around it. As soon as the paste had set the cell was sealed with a rosin or pitch composition.

The Gassner cell, though a great improvement over various other dry cells, did not meet with much favor because of its high internal resistance and low voltage. The highest current that could be obtained was about 6 amperes, and its voltage was about 1.3. Hence its use was limited to service requiring only small current drains.

DEVELOPMENT.

The advent of the gasoline engine greatly stimulated the development and production of the present type of dry cell. Ignition systems demanded a cheap portable cell able to recuperate after comparatively heavy current drains and by 1897 the manufacture of such cells had attained considerable volume. The present normal yearly requirements of dry-cell production include about 25,000 tons of high-grade manganese dioxide ore, an equal amount of carbon (petroleum coke and graphite) and 8,000 to 10,000 tons of sheet zinc, besides corresponding quantities of zinc chloride, ammonium chloride, paper, carbon electrodes, pitch, and sundry other substances.

METHOD OF MANUFACTURE.

In the modern American dry cell of the usual type, the negative pole is a cylindrical zinc can which also serves as a container. Several sizes of cells are on the market, the great majority, however, being about 2½ inches in diameter by 6 inches high (the so-called No. 6 cell). The inner surface of the zinc is lined with a special grade of absorbent paper, which acts as a reservoir for the electrolyte and as a diaphragm between the zinc and the positive pole. The depolarizing mass, called the "mix," is tightly tamped into the can, around a centrally placed carbon electrode, to about 1 inch from the top. This "mix" is composed of ground carbon (usually calcined petroleum coke and graphite) manganese dioxide ore, and the electrolyte. The mix with the carbon electrode, constitutes the positive pole. After the mix has been tamped, the paper is turned down over the mix; sand or sawdust is poured in, to a depth of about one-half inch, and the cell is sealed with a hot pitch composition. The object of the layer of sand is to provide an expansion chamber for the electrolyte and the excess gas, and also to provide a dry bed for the hot pitch.

CONSTITUENTS USED.

The sheets used for the cans are substantially pure zinc, prime western spelter being generally used, and are usually about 0.016 to 0.019 inch thick in the 2½ by 6 inch cells. Comparatively little of the zinc is used up during the service life of a cell. The mechanical strength to withstand filling and the fact that the can corrodes unevenly, in spots, patches, or streaks, must be taken into account when considering the necessary thickness. One of the problems of the dry cell manufacturer is to construct the cell so as to make this corrosion uniform.

Formerly, blotting paper was largely used for lining the cans, but at present most manufacturers use a special grade of pulp board. This is of ground wood pulp and sulphite pulp, and is 0.03 to 0.04 inch thick. The pulp board should be porous enough to allow the electrolyte to diffuse readily through it, but still retain the smallest particles of carbon and manganese; and it should be capable of absorbing several times its weight of water. Furthermore, it should obviously be free from metallic particles.

FUNCTIONS OF DIFFERENT CONSTITUENTS.

The depolarizing mass or "mix" is the vital part of the cell. To render the most efficient service its different components must be properly proportioned.

The ammonium chloride should be as pure as the usual chemically pure article. It should be substantially free from alkalies, sulphates, inert material, and heavy metals. When the cell is discharging, by virtue of the carbon, manganese dioxide, and zinc couple the ammonium chloride is split up into hydrogen, ammonia, and chlorine, which attacks the zinc can, forming zinc chloride. The reaction is: $Zn+2NH_4Cl=ZnCl_2+2NH_3+2H$. The liberated gases cause polarization.

Artificial graphite is generally used, several grades of it being made for dry cells. The graphite does not enter into the chemistry of the cell, merely serving, with the coke, to render the mix more conductive. The coke used is calcined petroleum coke—that is, residues remaining in petroleum stills. When raw it is practically a nonconductor. However, on being calcined at a high temperature it becomes denser and as good a conductor as the better grades of amorphous natural graphite.

The relative fineness of the graphite, coke, and manganese ore and their distribution should be so uniform as to make the mix approach a solid porous mass. The current should flow along radial lines from the carbon electrode to every point on the surface of the mix adjacent to the paper lining. The ideal condition is to have each particle of manganese coated with enough carbon to render it a good conductor but still porous enough to permit efficient depolarization, while the voids should be filled with the porous coke.

The zinc chloride should be as free as possible from heavy metals. Its function is to depolarize the ammonia, which it does by the formation of double salts of zinc and ammonium chloride. No exact information is available as to the reactions involved, but it appears that a slightly soluble double chloride of zinc and ammonium is formed as the end product. During the earlier part of the service life of a cell, inefficient depolarization of the ammonia is probably often the cause of failure under heavy drains. A cell that has been short-circuited or subjected to a heavy drain smells strongly ammoniacal when opened. Another cause of dry-cell failure is the formation of a highly resistant, nonporous crust between the paper lining and the zinc can, probably through the formation of the double salt mentioned or the formation of zinc hydrate.

ROLE OF MANGANESE.

The manganese serves to depolarize the hydrogen. The reaction involved is usually given as follows: $2H + 2MnO_2 = Mn_2O_3 + H_2O$. The manganese reacts almost instantaneously in depolarizing the hydrogen, very likely while the latter is still nascent.

CHARACTER OF MANGANESE ORE USED.[a]

CHEMICAL REQUIREMENTS.

Several factors determine the suitability of manganese ore for dry cells. The ore should have a high available oxygen content present in the form of pyrolusite (MnO_2), should have a minimum amount of iron, and should be free from copper, nickel, cobalt, arsenic, and other metals electronegative to zinc. Copper is particularly harmful. If these impurities are present in the electrolyte or insoluble compounds they do no harm other than as inert or poor conducting materials. If soluble, however, their solutions diffuse to the zinc can, where they are deposited, forming an electrocouple which causes local and useless corrosion of the zinc and consequent deterioration of the cell. When the cell is in service, the deleterious action of these impurities is greatly hastened.

---

[a] Storey, O. W., Determination of manganese dioxide in pyrolusite: C. F. Burgess Laboratories, Madison, Wis.

PHYSICAL REQUIREMENTS.

The physical properties of manganese ores influence their suitability for use in the dry cell. An ore should be somewhat porous to perform its function efficiently. A somewhat hard but porous ore is likely to give better results than a hard, dense ore, even though the latter is higher in available oxygen. In a dense ore the depolarizing reaction takes place only on the surface, whereas in a porous ore it can occur throughout the mass. Better service life is obtained from a dry cell containing rather coarsely ground ore, as this can hold more electrolyte than finely ground ore. Other factors of importance are the porosity of the ore and the fact that more contact resistance exists between particles of fine ore than those of coarse ore. Careful grading of the ore greatly influences the performance of a cell. The ore, therefore, should not be of an earthy nature like wad, as this mineral does not lend itself to efficient milling and grading.

Before the war the manganese ore used in dry cells was Caucasian pyrolusite. Common specifications called for material containing 80 to 85 per cent MnO_2 and less than 1 per cent iron. No particular attention was given to other ingredients, at least by most buyers, because of the purity and uniformity of the Caucasian ore. There are considerations other than the content of MnO_2 which determine the usefulness of manganese for depolarizer purposes, such as the screen analysis, hardness, density, and other physical qualities. Various manufacturers employ different specifications as to the screen analysis, a common specification being that the run of material shall pass through a 10-mesh or a 20-mesh screen. Some manufacturers specify the removal of the fine particles.

During the war manganese from many other sources was used in making dry cells because of the scarcity of the Caucasian ore. Most of these ores run lower in MnO_2 and higher in iron and have larger percentages of impurities, some of which are decidedly harmful. During the war users accepted material that ran 70 to 80 per cent MnO_2 and as high as 3 to 4 per cent iron. Users have found by experiment and manipulation how to get results with domestic ores and foreign ores other than Caucasian, closely approximating the results obtained with Caucasian.

An important source of manganese dioxide during the war was the ore from old dry cells, which was rejuvenated by processes generally kept secret by the firms employing them. Such processes doubtless contributed to the conservation of our high-grade domestic manganese ore.

In the manufacture of dry cells two classes may be recognized—the standard or so-called No. 6 cell, which is used for ignition, telephone, signal, and other similar purposes; and the small size, or flash-light type, which is used for portable lighting. The quantity of pyrolusite ores used in the standard No. 6 cell is far larger than that used in other sizes. In the manufacture of flash lights, which is growing rapidly, the higher grades of materials are required, such as 80 to 85 per cent ore that has been purified and also various grades of chemically prepared manganese dioxide and hydrates of manganese.

MANGANESE ORES IN THE CERAMIC INDUSTRIES.

GLASS MAKING.

Practically all the raw materials used in glass contain some iron, usually in the form of ferric oxide. The iron, when present even in small quantity, imparts to the glass a pale green color that increases rapidly in intensity as the iron content increases. If a colorless glass is desired, this green color must be removed by some decolorizer. Manganese, selenium, cobalt, and nickel are the most common decolorizers in use, and of these manganese has been most widely employed because it permits easy control of the color. In using selenium and nickel, the quantity must be carefully controlled, but these latter substances are desirable, especially in window and plate glass, because glass decolorized with manganese often changes to a pink color on exposure to the light. A decreasing quantity of manganese is being used by makers of tank glass, and its place is being taken by selenium.

The quantity of manganese used varies considerably, depending on the character of the glass, the method of its manufacture, the iron content of the raw materials, and the character of the manganese ore used. Each manufacturer has his own ideas on this subject. The quantity used is figured in terms of pounds of manganese dioxide per 1,000 pounds of sand, which constitutes 50 to 75 per cent by weight of the entire batch. The temperature employed in the glass-making process helps to determine the quantity of manganese dioxide used, for because of volatilization the higher the temperature, the more manganese is necessary. The maximum limit is 10 to 15 pounds of manganese dioxide per 1,000 pounds of sand, and the minimum may be 2 to 2½ pounds.

CHEMISTRY OF USE OF MANGANESE IN GLASS-MAKING PROCESS.

Compounds of manganese, when other coloring ingredients are absent, produce pink, purple, and violet hues according to the chemical nature of the glass. Manganese dioxide neutralizes the green color caused by iron compounds. Used in excess, it imparts an amethyst tint, and when used in considerable excess, the color is so dark as to appear black.

The neutralization of the iron tint by manganese dioxide is explained by some chemists on a physical, and by others on a purely chemical basis. The green tint is due to the presence of ferrous silicate. Some chemists think that this green compound is oxidized to the ferric silicate, which has an almost imperceptible pale straw-yellow color. According to this view, the oxidizing agent used must not completely decompose at high temperatures, and manganese dioxide seems to be the most available compound fulfilling this condition. At red heat, the dioxide loses one-third of its oxygen, leaving the tetraoxide (Mn_3O_4) which, at still higher temperatures, is an oxidizing agent.

Red lead and other oxidizing agents have not this decolorizing power. Hence some chemists have thought that the result is not due to oxidation, or chemical reaction, but is purely physical. It is possible, however, that other compounds may lose their oxygen at too low temperatures to be effective as oxidizing agents.

SPECIFICATIONS FOR MANGANESE ORE USED IN MAKING GLASS.

Before the war, the ordinary specifications for manganese ore used in glass making were 85 to 90 per cent manganese dioxide and less than 1 per cent metallic iron. Outside of these two ingredients, each manufacturer has his own requirements. Special glasses may require ore carrying more than 90 per cent manganese dioxide and less than 0.5 per cent iron. The higher the manganese content and the lower the iron, the better the ore is for glass making. In general, the grades of ore are similar to those used in making dry cells. Obviously, siliceous pyrolusite is not objectionable but carbonaceous pyrolusite is.

Manganese ore for glass making is sold in powdered, granulated, or lump form. There are objections to the lump form because of the time required to melt it into the batch. Powdered ore is used principally when the batch is melted in pots; lump, or granular ore is used when melting is done in tanks.

Before the war, high-grade pyrolusite for glass making and other chemical purposes was imported from Russia, Saxony, Japan, Nova Scotia, and other foreign countries. As the war progressed, such ore became scarce and, as a consequence, specifications were relaxed and low-grade ores were purchased. During the war some excellent domestic ore was developed, which found a ready market.

OTHER CERAMIC USES.

Another use for manganese ore in glass making has developed in the last few years, namely, for producing black glass used for ornamental purposes. About 3 per cent of ore is added to the batch in making this opaque glass.

Pyrolusite is added to the constituents of glazes and enamels to produce purple tints. Black enamels are those containing manganese. Manganese oxide is also used in brick making.

USE OF MANGANESE SALTS IN DRIERS.

DEFINITION.

Driers are substances, generally metallic oxides or their compounds, that are added to linseed or other drying oils at high or low temperatures to make them capable of readily absorbing oxygen from the air, or of drying by its action. Some chemists consider the action to be catalytic, the manganese compound acting as a catalyser or carrier of oxygen. The principal manganese compounds used as driers are: Manganese sesquioxide (Mn_2O_3), pyrolusite (MnO_2), also known in the trade as dioxide, binoxide or peroxide, manganese hydrate, sulphate, borate, resinate, linoleate, oxalate, and possibly other salts. Certain of the corresponding double salts of manganese and lead are often used.

Some persons claim that pyrolusite is now little used in driers because the manufactured hydrate, on account of its purity, gives better results. This claim does not agree with statements made by dealers in the trade. Of the various substances named above, each acts in a way peculiar to itself. These driers are added only in small quantities, usually less than 0.5 per cent.

MANGANESE DIOXIDE.

Manganese dioxide, extensively used as a drier, is marketed in two forms, the natural and the artificial. The natural mineral, pyrolusite, is simply ground to a powder with water and then dried. The mineral is essentially a peroxide, a class of substances containing more oxygen than is required to satisfy the valence of the metal present. This extra oxygen is loosely combined and readily enters into combination with oxidizable bodies. This feature in the composition of manganese compounds makes them useful in oil boiling, because the oxygen combines with the oil, oxidizing it, while some of the manganese dissolves and forms a compound with the linoleic acid of the oil. In consequence of this action manganese compounds are powerful driers. The quantity of manganese dioxide added in the process of boiling is small, not more than a quarter of a pound to a hundred weight of oil to get the best results. The use of the black dioxide, however, tends to make the oil dark.

MANGANESE SULPHATE.

The methods of preparing this compound and its uses are described on page 24. Rather less than one-half a pound is added to each hundred weight of oil or paint. Oil boilers use it largely as a dryer of pale boiled oils.

MANGANESE BORATE.

Manganese borate is perhaps the least objectionable of all the manganese salts used as drying agents, although the black oxides are more used. To make the borate, 1 part of manganese sulphate is dissolved in 10 parts of distilled water and a little of this solution is added to some soda solution to determine whether any iron is present. If pure, the manganese sulphate solution is added to the hot borax solution as long as a precipitate forms. The precipitate is filtered, washed with hot water, and dried.

MANGANESE RESINATE.

Resinates, formed by the combination of resin or rosin with certain metallic oxides, are much used in making varnish and paint and with the exception of the linoleates are most readily soluble in linseed oil. Manganese resinate is made as follows: Soda ash is dissolved in water and the solution is boiled with steam, the proper proportion of light coarsely powdered rosin is added, and then more soda ash. The clear solution is run into a clear manganese chloride, the resinate separating as a white flocculent precipitate, which is filtered, washed, and dried.

MANGANESE LINOLEATE.

Manganese linoleate is made by pouring a solution of a soap, made by boiling linseed oil and caustic soda into a solution of manganese chloride or manganese sulphate. The dark-brown plaster-like mass is liable to oxidation. When exposed to air, the surface becomes covered with a hard, rather insoluble, protective skin. The material should, therefore, be kept in tightly closed vessels. It acts both as a bleaching agent and a drier for linseed oil. One pound, mixed first with 5 pounds of linseed oil and the whole poured into 10 gallons of linseed oil at 250° F. gives a good drying oil.

The double resinates and linoleates of manganese and lead are also in use.

MANGANESE OXALATE.

Manganese oxalate is prepared by precipitating manganese salts with oxalate of soda or potash or by treating manganese hydrate or carbonate with oxalic acid. One of the advantages of the oxalate is said to be that it decomposes during oil boiling. The manganese dissolves in the oil in combination with linoleic acid, and the oxalic acid is decomposed with evolution of carbonic acid. One-fourth to one-half a pound may be used per hundred weight of oil.

USE OF MANGANESE IN MISCELLANEOUS CHEMICALS.

MANGANESE CHLORIDE.

Manganese chloride ($MnCl_2$) is prepared by the action of hydrochloric acid on the dioxide, chlorine being a by product of the reaction. It is a rose red, deliquescent salt used in dyeing cotton cloth a manganese brown or bronze. The fabric to be dyed is soaked in the salt solution and passed through a caustic alkali, whereby manganese hydroxide is precipitated in the fabric and on subsequent oxidation turns brown. The material thus treated may be used also for subsequent dyeing by anilin black.

MANGANESE SULPHATE.

Manganese sulphate ($MnSO_4$) may be prepared on a large scale from the black dioxide by heating to redness with ferrous sulphate and subsequently extracting with water. The salt forms pink crystals which are readily soluble in water and are used in calico printing and in porcelain painting. It is also used as a drier for pale oils or for conversion into the oxalate or borate which are used for the same purpose.

MANGANESE PERSULPHATE.

Manganese persulphate, $Mn(SO_4)_2$ is prepared by the electrolytic oxidation of manganous sulphate ($MnSO_4$) and forms a black substance that can be obtained in solution only in the presence of sulphuric acid. It is used as an oxidizing agent in the manufacture of organic products.

POTASSIUM PERMANGANATE.

Potassium permanganate ($KMnO_4$) is prepared industrially by mixing a solution of caustic potash (KOH), specific gravity 1.44, with powdered manganese dioxide and an oxidizing agent, such as potassium chlorate. The mixture is boiled and evaporated, and the residue is fused in crucibles and heated until it has a pasty consistency. The potassium manganate (K_2MnO_4) thus obtained is dissolved by boiling with much water while a current of chlorine, carbon dioxide, or ozone is passed through the liquid. Potassium permanganate separates in crystalline form from concentrated solutions even in the presence of the caustic potash formed during the reaction, and is separated from the dissolved substances in a hydroextractor.

The permanganate is used for preserving wood; it is also used for bleaching textile fibers, by immersing them for a time in an aqueous solution of it and then dissolving the manganese dioxide with sodium disulphite. The permanganate is an energetic disinfecting and oxidizing agent and is used for purifying various gases.

USE OF MANGANESE IN MANGANESE BRONZE.

GENERAL REMARKS.

During the last twenty years manganese bronze has been widely used. The requirements of marine construction, of mining machinery, and wherever corrosion has presented a serious problem, have created a demand for a nonferrous metal to replace steel. Probably the most popular of such substitutes has been manganese bronze.

Manganese bronze made its first appearance about 1876. Its name is somewhat misleading for the alloy contains only a small percentage of manganese. Indeed, it is simply a brass to which have been added by proper methods of alloying, small quantities of aluminum, iron, or manganese, for the purpose of strengthening the alloy and making it denser and closer grained than the average yellow-brass casting.

FUNCTION OF MANGANESE IN MANGANESE BRONZE.

Manganese bronze should not contain much manganese, in fact not more than 0.05 per cent in high-grade bronze. Consequently the consumption of manganese in such alloys is small. The object of the manganese is not so much to act as an ingredient of the alloy, as to serve as a carrier of the iron necessary to insure the required strength and elastic limit. The manganese serves one purpose only, to introduce the iron, for without the manganese the iron would not alloy with the copper. Usually the manganese is added in the form of ferromanganese. If added in large quantities it hardens the alloy, but not nearly as much as iron; such an addition also lowers the elastic limit. Aluminum imparts a good sand-casting quality to the bronze.

Two grades of manganese bronze are now in common use. One is used for rolling into sheets, or drawing into wire or tubes, and for forging. This grade contains no aluminum, and has slightly less zinc than the other and can not be cast in sand. The second alloy is used for sand casting and is the one employed in making propellers and other common appliances.

The method of making manganese bronze—that is, the materials used, the methods of combining them, and the process for casting is discussed by Sperry [a] and will not be discussed here.

McKinney [b] describes a process of manufacturing manganese bronze wherein is used, instead of virgin metals and raw materials of the highest purity, by-products and scrap. The methods described are, therefore, timely.

The composition of manganese bronze is as follows:

Composition of manganese bronze.

Constituent.	Per cent.
Copper	57.00–59.00
Zinc	38.00–40.00
Iron, manganese, aluminum, and tin	.25– 1.00
Lead	.10– .50

Evidently there is no particular need of using high-grade materials, provided the finished product is properly refined. Among the low-grade materials suggested for use in making manganese bronze are the following: Skimmings from the foundry, especially skimmings and dross ordinarily recovered from brass rolling mills or cartridge-case plants; zinc dross recovered from galvanizing plants; aluminum turnings that are generally unrecoverable without serious loss and deterioration of product through oxidation, and other by-products and scrap metals that ordinarily are not usable in foundry practice as remelting scrap.

---

[a] Sperry, E. F., Manganese bronze and its manufacture: Brass World, vol. 1, December, 1905, pp. 399–406.

[b] McKinney, P. E., Manganese bronze: Am. Inst. Min. Eng. Bull. 146, February, 1919, pp. 421–425.

High-grade manganese bronze can not be made from the above raw materials on a small scale and thus manufacture in crucible furnaces is excluded. A reverberatory furnace or other equipment with which a bath of considerable proportion may be employed is necessary. McKinney [a] discusses a typical charge, the materials being melted in the presence of charcoal with salt as a flux.

USES OF MANGANESE BRONZE.

The most important use of manganese bronze is in propeller blades. A strong, tough alloy is necessary which will resist the action of sea water. The blades are made thin to save weight.

---

[a] McKinney, P. E., Manganese Bronze: Am. Inst. Min. Eng. Bull. 146, February, 1919, pp. 421-425.

CHAPTER 3.—PROBLEMS INVOLVED IN THE CONCENTRATION AND UTILIZATION OF DOMESTIC LOW-GRADE MANGANESE ORE.

By Edmund Newton.

INTRODUCTORY STATEMENT.

In the past, the steel industry of the United States has depended almost wholly on imports for its supplies of manganese. Many of the important domestic sources yield ores that in their natural condition contain less manganese than the foreign ores the steel industry has been accustomed to use. To make these domestic ores available, therefore, they must be concentrated or practice in the steel industry must be modified.

Roughly, 25,000 tons of high-grade manganese ores is used annually for dry batteries, for chemical purposes, and in other minor ways, and approximately 750,000 tons is required for making steel.

By present practice every ton of steel takes an average of about 15 pounds of metallic manganese, which generally is added to the steel in the form of an alloy. The standard alloys are 80 per cent ferromanganese and 20 per cent spiegeleisen. During the year 1917, 286,000 tons of ferromanganese and 193,291 tons of spiegeleisen were made in this country, the former largely from imported ores; and 45,381 tons of ferromanganese was imported. The metallic manganese represented by these alloys was 304,000 tons, being roughly the product of 800,000 tons of high-grade ore and 345,000 tons of low-grade ore.

There is an abundance of low-grade ore in this country suitable for the manufacture of spiegel, but higher grade ore is necessary to make ferromanganese. For this reason the concentration of domestic ore presents a field for constructive and practical research.

MANGANESE DEPOSITS IN THE UNITED STATES.

Before the war manganese ore was mined in relatively small quantities in the Appalachian region, which includes parts of Virginia, Tennessee, and Georgia, and in Arkansas, but in consequence of higher ore prices because of the rise of ocean freight rates, manganese mining has been undertaken in Montana, California, Arizona, New Mexico, Nevada, Utah, and Minnesota as well as in the Appalachian region.

Data now available indicate that in this country deposits of high-grade manganese ores are usually small, but some deposits of ore lower in manganese and higher in iron are of considerable size. In the aggregate, the total quantity of manganese-bearing material is relatively large, but the difficulty of mining small deposits of the better grades of material and the seeming undesirability of low-manganese alloys in the steel industry, make the outlook for large production of manganese in this country uncertain.

As regards the geologic origin of the majority of the manganese and manganiferous iron ores in this country, Harder[a] states that they are largely the result of secondary concentration. Most of the ores of the Eastern United States, Arkansas, the Lake Superior region, Leadville and other silver districts, and of western California are of this type. The rhodonite and rhodochrosite in the unoxidized parts of the silver veins at Butte, however, are primary concentrations derived from igneous intrusion. The ores of northern Arkansas are largely reconcentrations from low-grade secondary deposits, derived by decomposition of crystalline rocks, and the California ores are concentrations within chert lenses of material originally present in a disseminated form.

Manganese-bearing materials of the United States may be roughly classified as follows:

1. Manganese ore proper.
2. Manganiferous iron ore.
3. Miscellaneous material:
    (*a*) Manganiferous silver and lead ore.
    (*b*) Zinc residuum from manganiferous zinc ore.

Manganese ore, as now defined by the trade, is material that contains more than 35 per cent manganese and is suitable for the manufacture of 70 per cent ferromanganese. Manganiferous iron ore contains less manganese and more iron. In general, the iron predominates, but there is no hard and fast line of demarcation between manganese ore and manganiferous iron ore. Manganese and iron are so closely associated in nature that all gradations from low-manganese, high-iron ore, to high-manganese, low-iron ore may be found in various deposits or in the same deposit.

Manganiferous silver ore is similar to manganiferous iron ore; it carries enough silver to make it valuable for that metal. Commercial considerations alone control the balance between the manganese or the silver value.

Zinc residuum is a by-product of the smelting of zinc ores from Franklin Furnace, N. J., which contain considerable manganese. After the zinc is removed the remaining product, called residuum, has nearly the same composition as natural manganiferous iron ore, and for years it has been smelted to spiegeleisen.

---

[a] Harder, E. C., Maganese deposits of the United States: U. S. Geol. Survey Bull. 427, 1910, p. 4.

In general it may be said that the domestic manganese ores consist of an aggregate of minerals of manganese and iron with various impurities, such as silica, alumina, lime, and magnesia and accessory constituents, such as phosphorus, sulphur, silver, lead, and zinc. The determining characteristics of the manganese minerals and their relation to impurities are discussed in subsequent pages.

In the steel industry manganese is chiefly used in the form of alloys. A less important use is for increasing the manganese content of pig iron to give particular grades, as foundry irons, or to assist in metallurgical operations of certain steel-making processes, as in the basic open-hearth process. The alloys of manganese generally used in this country are ferromanganese, formerly containing 80 per cent, but now 70 per cent, metallic manganese, and spiegeleisen, having 15 to 20 per cent metallic manganese. The rest of these alloys consists principally of iron with small quantities of carbon, silicon, and phosphorus.

During the last few years ferromanganese has been gaining popularity among steel manufacturers, with spiegeleisen declining proportionately. Until recently approximately nine-tenths of the metallic manganese used in the steel industry was in the form of the standard 80 per cent alloy. Ferromanganese, or "ferro," as it is usually called, is easier to use in steel making than alloys containing smaller quantities of manganese, as the required quantity of that metal is contained in a smaller bulk.

The difficulty of obtaining ores suitable for the production of "ferro" has led to a consideration of the possibility of using what may be called intermediate alloys with manganese contents varying between 20 and 80 per cent. In the electric furnace certain alloys can be made with a relatively large content of silicon, in addition to the manganese and iron. The extent to which such alloys may satisfactorily be used in steel manufacture is not alone a technical or economic problem, but is largely controlled by the human element and the unwillingness of steel masters to deviate from long-established practice.

Phosphorus is undesirable in finished steel. In the manufacture of manganese alloys all the phosphorus contained in the ore will be recoverable in the alloy and will enter the steel when the alloy is used. It is permissible, however, for an alloy high in manganese to contain more phosphorus than one low in manganese, for less of the former alloy is needed to carry a given quantity of manganese. It is interesting to note that when 17 pounds of 80 per cent ferromanganese is added to a ton of steel the alloy may contain 1.33 per cent phosphorus and yet increase the phosphorus content of the steel by only 0.01 per cent. To add the same amount of manganese in the form of spiegel, the alloy may contain 0.33 per cent phosphorus and increase the phosphorus content of the steel 0.01 per cent.

For many years manganese alloys have been made chiefly in the blast furnace, although recently certain plants have produced them in the electric furnace. Blast-furnace practice on manganese alloys is generally similar to ordinary pig-iron practice, but there are high metal losses, principally in the slag, by volatilization and as flue dust. The amount of manganese contained in a given weight of slag may be partly controlled by furnace manipulation, but it is evident that the total amount of manganese lost in this manner is directly proportional to the "slag volume."

The results of increased slag volume and greater loss of manganese are cumulative and rather serious. More ore must be used per ton of alloy produced. This additional ore carries more slag-forming constituents. More coke is required to melt it, which in turn tends to produce more slag, and increased slag volume cuts down the daily output of alloy from the furnace. The greater loss of manganese decreases the ratio of manganese to iron in the alloy, so that unless proper allowance is made the alloy will be below the standard grade and therefore subject to a penalty by the purchaser. Not only will the alloy sell for less money, but the decreased daily output of the blast furnace will lessen the total profits.

A manufacturer of manganese alloys endeavors to protect himself against these decreased profits and adjusts his schedules and penalties for ore purchase with that end in view. Although he endeavors to equalize the effects of poor ores, the alloy producer would prefer to buy better ores at correspondingly higher prices.

CONCENTRATION OF DOMESTIC LOW-GRADE MANGANESE ORES.

The comprehensive term "concentration" as here used is intended to cover the improvement of low-grade ore by any suitable means preliminary to smelting. The requirements of metallurgical practice control the classification of manganese ore as low grade and high grade. Thus the term low grade may refer to a low content of manganese with respect to iron or to large quantities of nonmetallic impurities. The detrimental effects in metallurgical practice and the resulting penalties are the incentive for attempts to improve the ore or raise its grade before smelting.

FACTORS CONTROLLING THE POSSIBILITIES OF CONCENTRATION.

In order to interpret properly the possibility of concentrating at a profit any type of manganese-bearing material, many technical and economic factors must be considered. For a particular property, district, or class of material it is necessary to obtain data on these factors:

1. Character and size of the deposit,
2. Conditions affecting mining and marketing,
3. Character of ore material as affecting the possible improvement of grade,
4. Metallurgical value of crude ore and possible concentrate,
5. Commercial considerations.

SIZE AND CHARACTER OF DEPOSIT.

Obviously a large deposit of low-grade manganese ore would warrant considerable experimental work in order to determine satisfactory methods of treatment. Conversely, if a particular type of ore occurred in only one deposit of only a few thousand tons, the value to the industry of the product from such a deposit would be relatively small, even if concentration were feasible. Therefore, no undue proportion of time should be devoted to a concentrating problem that although of considerable technical and individual interest, gives little promise of furnishing any considerable part of the industry's needs.

The mineralized mass must be of such size and character as to justify the expenditure of money in its development and beneficiation and to promise return interest on the investment proportional to the risk taken. This factor of cost is of vital importance, and it is feared that under the stimulus of production incident to national needs during the war sound business principles have at times been temporarily forgotten.

CONDITIONS AFFECTING MINING AND MARKETING.

Not only must the quantity and character of material available be considered, but the natural factors controlling the mining methods, the transportation facilities, and the marketing of products must be carefully studied.

The manganese deposits of the United States, although widely scattered and comparatively small, may nevertheless be mined by relatively simple and therefore cheap mining methods. The mines are for the most part shallow, and extensive development and elaborate equipment are not necessary. Intricate problems of ventilation and drainage do not have to be solved, and if all operations are competently directed common mine labor will generally suffice. Small deposits mean short life, the ore is mined out rapidly and more or less crude mining practice prevails. The cost of mining will, however, be more or less governed by the necessity of selective mining, which in turn is determined by the variability of the ore, the feasibility of economic concentration, transportation facilities, and distance from a consuming center.

All these factors must be coordinated and their combined influence carefully studied before intensive production from individual properties begins. Concentration by eliminating waste may yield a product desired by the steel industry, but its cost may be prohibitive. Discarding waste may enable the producer to offset excessive freight rates, but geographic isolation will invariably handicap an enterprise.

Foreign ores will always find a market in the United States because they come from larger and more uniform deposits, are mined with cheaper labor, and the ocean freights are lower than rail.

CHARACTERISTICS OF ORE AFFECTING BENEFICIATION.

CHARACTER OF MANGANESE MINERALS.

Although many minerals contain manganese, only a few are commercially important. Usually it is rather difficult accurately to identify the manganese minerals in domestic oxidized ore. Several minerals may be intimately associated, and one may have been formed by alteration of another. The hardness of the individual minerals varies widely. Pyrolusite is soft and may be readily pulverized between the fingers. Hence difficulty might be expected in attempts to recover this mineral by the common processes of wet concentration. The other minerals are harder, but usually brittle. Although the character of the individual minerals is important, the association of the several manganese minerals with various gangue materials often has a more important bearing on methods of concentration.

IMPURITIES ASSOCIATED WITH MANGANESE MINERALS.

Wherever manganese ore is mined on a commercial scale, the product of the mine always contains impurities. Some of these are obvious on inspection; others may require chemical analysis for their determination. The impurities associated with manganese minerals may be classified as (1) those derived from associated rocks, or rocks partly replaced by manganese-bearing solutions, (2) those associated with the manganese in solution, and deposited simultaneously, and (3) those chemically combined with manganese in the mineral.

From the viewpoint of the metallurgist, all are impurities, and must be removed either before or by metallurgical treatment.

For convenience, the common impurities in manganese ores may be classified according to certain general physical and chemical principles, as follows:

1. Metallic. Iron, lead, zinc, silver, and in some ores, nickel, copper, and tungsten.
2. Gangue. "Basic" lime, magnesia, baryta, "acid" silica, and alumina.

3. Volatile. Water (atmospheric moisture and molecular water), carbon dioxide, and organic matter.

4. Miscellaneous. Phosphorus and sulphur.

The chemical behavior of these impurities affects metallurgical operations, and the physical form in which they occur controls the possibility of removal previous to smelting, and the choice of methods for accomplishing such removal.

The proportion of manganese to the useless or harmful constituents of an ore determines the value and desirability of the ore.

The presence of appreciable quantities of any impurity means that more ore must be mined and smelted in order to produce a given weight of manganese alloy. Some impurities are more detrimental to alloy manufacture than others.

Metallic impurities, of which iron is the most common, are usually reduced in smelting and are retained by the alloy. The quantity present naturally affects the character of the alloy produced, which in turn controls its desirability for use in steel manufacture. Other metallic impurities occur usually in such small amounts that they are not detrimental to the resulting alloy. Zinc is an exception. This metal is largely volatilized in smelting, and if it is present in appreciable quantities its fume condenses as oxide in the hot-blast stoves and may hinder furnace operation. Unless the furnace-top gases are washed, the stoves must frequently be cleaned, with consequent loss of time. When the price of zinc is high, the zinc oxide recovered from the stoves yields a substantial sum.

Silver, from the standpoint of the steel manufacturer, is neither detrimental nor advantageous to manganese alloys. The silver content of a manganese alloy has no value; consequently no credit is allowed the miner for silver contained in an ore to be used for manganese-alloy manufacture. In some ores the silver content is such that the ore has greater value for the lead smelter. The manganese then acts as a flux and the silver may be recovered by purification of the lead.

The gangue impurities classed as basic and acid may also be called "slag-forming impurities." In smelting, these impurities must be fluxed to form slag. Slag is usually considered a waste product, but it has important metallurgical functions, and just enough slag must be present for performing these functions properly and economically. An excess of slag must be avoided. In manganese-alloy manufacture the slag contains more or less manganese, the quantity of manganese thereby lost being dependent on the basicity of the slag, the temperature, and the slag volume. The first two factors control the quantity of manganese in a given weight of slag, and it is obvious that the greater quantity of slag will result in a greater loss of manganese. A large slag volume will rapidly decrease the daily alloy output of a blast furnace. The overhead charges must be distributed over a smaller daily tonnage of alloy and the profit from the sales for a given unit of time will decrease. It follows that more than the quantity of slag required to provide for the metallurgical functions is highly undesirable.

Silica is usually the predominant slag-forming constituent in domestic manganese ore. Some silica is reduced to the metallic state in smelting and is recovered in the alloy as silicon, but the larger part must be fluxed with lime, magnesia, or other bases to form slag. Manganese-alloy slags should be basic; hence a larger quantity of slag will be produced from an ore with acid gangue than in normal iron blast-furnace practice. Alumina is a slag-forming constituent and although usually classed with silica, it acts somewhat differently in the blast furnace. Brazilian ores are notably high in alumina, but most domestic ores contain relatively small quantities.

Lime, magnesia, and baryta in an ore are also slag-forming constituents, but they combine with the silica and alumina present and thereby reduce the quantities of those bases necessary in the form of limestone or dolomite for the furnace charge. Baryta is not common as a gangue mineral. It is not as strong a base as either lime or magnesia. While these constituents offset the metallurgical effects of silica or alumina, as regards evaluating an ore, they represent weight, and if the ore must be transported a considerable distance to the point where it is smelted, it is doubtful whether their value as bases would equal the additional freight charge. Limestone can generally be obtained at low cost close to the smelter.

Volatile impurities are removed from the top of a blast furnace largely by the surplus heat. It is desirable, however, in order to reduce the loss of manganese to keep the top of a manganese-alloy blast furnace cool. Volatile compounds are not particularly detrimental to smelting. When carbonate ores are being treated the case is somewhat different. Some metallurgists claim that in treating rhodochrosite ores the ratio of carbon monoxide to dioxide in the furnace gases is disturbed, which has a detrimental effect on the reduction of the oxides in the upper part of the furnace. It has also been suggested that the carbon dioxide driven off combines with carbon of the coke, forming carbon monoxide in the upper part of the furnace, and thus increases the coke consumption. Definite data are not available on these points.

In practice, all the phosphorus in the ore mixture and that contained in the coke and limestone is recovered in the resulting alloy. The permissible quantity of phosphorus in an alloy (the proportion that does not produce detrimental effects when the alloy is added to steel) has not been definitely determined. The higher the manganese content of an alloy, the larger the proportion of phosphorus that may safely be contained. Ordinarily steel makers desire as large a margin of safety as possible, and therefore have specified that the phosphorus in an ore shall not exceed a certain percentage.

Sulphur is usually present in small quantities in oxidized manganese ores, but in the primary rhodochrosite ores of Butte and other parts of the West there may be considerable quantities of sulphides of iron and zinc. Sulphur is not a serious factor, as the conditions of blast-furnace operation when making manganese alloys are such that sulphur combines with manganese or lime and is readily retained by the slag, only traces entering the alloy.

Knowing the effect of impurities in manganese ores on blast-furnace practice, the methods of eliminating them may now be considered. Ore dressing deals with the problems of separating deleterious or useless materials from the more valuable minerals, thereby raising the grade and reducing the quantity of the concentrated product. To accomplish this, it is essential that the physical and chemical characteristics of the ore be determined. These factors are conditioned largely by the type of deposit from which the ore is mined. As types of ore, entirely disregarding genesis, we recognize:

1. Rhodochrosite and rhodonite; carbonate and silicate ores, deposited in fissure veins or replacing original rocks.
2. Nodular ores, accretions of manganese oxide in soft plastic clays.
3. Manganese oxides deposited in small fissures or fracture planes, as braccia fillings, or as more or less impure beds.
4. Manganese oxides occurring as infiltrations, deposited in minute pore spaces, as partial replacements, or otherwise intimately mixed with the rock or gangue.

In the first class of deposits, the principal gangue impurity is silica, although sulphides of silver, lead, zinc, and iron are often found in appreciable quantities. The silica occurs both as quartz and chemically combined in rhodonite ores. In the carbonate ore, the carbon dioxide may be removed by calcination, thus effecting a concentration of manganese, reducing the bulk, and lowering the freight rate per unit of manganese contained in the original ore, but rhodochrosite decrepitates strongly when heated to a temperature where the oxide is formed, tending to produce an excessive amount of fines which is undesirable in practice. The breaking up of the particles by calcination will isolate some of the free silica which on account of the larger sized particles might be screened out. The sulphide minerals may occur in such quantity that it is desirable to remove them by gravity methods of separation.

In the second class of deposits, the nodules are of variable size and usually high in manganese. They do not appear to be contaminated internally with the inclosing material. The clays are soft, whereas the nodules are generally hard. This type is common in the Appalachian region of the United States. The clay may be separated from the manganese nodules by log washers, followed where necessary, and where the size of the deposits warrants the installation, by picking belts, crushers, screens, and jigs.

In deposits of the third class, the manganese minerals, although closely associated with the inclosing rock, are generally not contaminated by it and may be relatively pure. The method of treatment varies with the size of the manganese particles and the hardness of the rock, but does not differ essentially from the treatment of the second class. If there is little or no clay, the log washer will be omitted, and crushing, screening, jigging, and possibly tabling will make up the concentrating process.

If the manganese mineral is largely pyrolusite, and therefore friable and soft, crushing may produce an excessive proportion of fines difficult to recover by gravity or water methods of separation. If, however, the manganese mineral is hard, dense, and massive, and the inclosing rock is more friable, the problem is simpler. When the specific gravity of the minerals and the gangue approximate each other, wet concentration is difficult unless the particles differ decidedly in size.

Obviously the association of gangue materials with the desired mineral in ores of the fourth class is so intimate that the finest crushing imaginable will not permit separation by mechanical means. To this type the siliceous manganese ores of the Western States may be assigned. Ore-dressing tests have conclusively shown that where the silica is chemically combined with the manganese or where colloidal silica envelops the manganiferous particles, any wet process or gravity concentration will not give the desired results.

CONCENTRATION PROCESSES.

This paper does not describe in any detail actual ore-dressing or concentration practice. It is axiomatic that small deposits or mines of questionable life, as determined by tonnage and markets, do not warrant elaborate plants or the adoption of intricate beneficiation processes. A general classification of methods applicable to the manganese industry is given below. The processes mentioned are all preliminary to the greater and final concentration of the desirable elements in the blast furnace.

I. Simple methods of concentration.
    (*a*) Selective mining.
    (*b*) Hand picking.
    (*c*) Jigging.
    (*d*) Screening.
    (*e*) Log washing.
    (*f*) Water classification.
    (*g*) Roughing-table treatment.
    (*h*) Slime-table or vanner treatment.
    (*i*) Pneumatic separation.
    (*j*) Combination of two or more of the above methods.

II. Complex methods.
  - (*a*) Magnetic separation.
    - (1) Without preliminary thermal treatment.
    - (2) With preliminary thermal treatment
  - (*b*) Electrostatic separation.
  - (*c*) Hydrometallurgical processes.
    - (1) Leaching with various acids. Precipitation by chemical substances.
    - (2) Leaching with various acids. Precipitation by electrolysis.
    - (3) Leaching with various acids. Evaporation of solution and heat treatment in rotary kiln.
  - (*d*) Preliminary thermal processes.
    - (1) Drying, to remove hygroscopic moisture.
    - (2) Calcining, to remove carbon dioxide or combined water.
    - (3) Agglomerating fine concentrates, to make them desirable for blast-furnace use.
    - (4) Volatilizing manganese at high temperatures in the presence of constituents that form readily volatile compounds.
    - (5) Direct reduction of oxides by carbon, under temperature control.
  - (*e*) Miscellaneous processes.
    - (1) Flotation.
    - (2) Use of heavy solutions.

There are many standard machines for the concentration of ores, but it is unwise to think that a certain machine will accomplish the necessary result on manganese-bearing materials. As the character of the manganese materials varies greatly in different districts, it is more logical to determine in detail physical, as well as the chemical, characteristics of the material. When such preliminary study has shown the nature of the impurity, and its relation to the manganese mineral, it is easier to outline a suitable method of treatment. The flow sheet, however, must be determined by experiment.

COMMERCIAL CONSIDERATIONS REGARDING BENEFICIATION.

If the technical possibilities of beneficiating any particular ore are favorable, it is then necessary to ascertain whether such an operation on a commercial scale would yield a reasonable profit. The cost of the plant and its installation must be justified either by the available ore in the deposit, or by the length of time during which the profit could be made. The amortization of capital and the interest on the investment must be included in the estimation of cost.

The effect of concentrating an ore is not always clearly appreciated. Concentration implies that an improvement of metallic content is made by the intentional elimination of impurities, but there is always a loss of the valuable mineral itself. When the grade of the product is increased, its weight is decreased. In other words, 2 to 25 tons of crude manganese-bearing material may be required to produce 1 ton of high-grade concentrate. The income results from the sale of the smaller quantity of concentrate, but chargeable against this will be the cost of mining some tons of crude ore, the cost of treating the ore, the freight to market, and the special overhead charges. Concentration, however, may be necessary to make the material marketable at all.

ESTIMATED COST DATA.

In order to illustrate the above points a little more clearly, the essential economic factors have been applied to three ores from California.

It is estimated that the three ores were obtained from desposits of such size that it would be possible to mine 75 tons of crude ore a day. In the table below the figures from actual concentrating tests have been combined with estimates of the cost of mining and treatment. The mining-cost figures are those reported by one of the three mines for the season of 1917.

A rough estimate of the cost of a small concentrating plant capable of treating 75 tons of crude ore a day is $20,000. It is assumed that this amount must be charged off in a year of 300 operating days. Instead of the amortization rate per ton being actually computed, the $20,000 and the interest charge of 7 per cent are distributed over the annual production of concentrates. The interest charge at 7 per cent is also distributed over the total tonnage of concentrates.

Royalty is considered to remain at $1.50 a ton of crude ore, even if the material is concentrated. Based on concentrates, it therefore increases proportionally to the ratio of concentration.

The value of crude ore and concentrates has been computed on the basis of the schedule approved by the War Industries Board in May, 1919. These prices no longer hold, and new computations should be made on the basis of existing prices. The methods to be followed are, however, indicated by the examples given. A freight rate of $11.20 per ton has been used from California points to Chicago on a long ton of 2,240 pounds.

All figures in these estimates are based on dry analyses and on a long ton of 2,240 pounds. Because of the length of haul, it is assumed that moisture would be negligible in both crude ores and concentrates. The figures follow in table 9.

TABLE 9.—*Estimated cost data applied to three California ores.*

MANGANESE ORE 1009.

	Fe	Mn	SiO_2	P
Analysis of crude ore, per cent	2.55	40.01	28.91	0.047

Concentration results:	Crude ore.	Concentrates.	Tailings.
Per cent by weight	100.00	50.55	49.45
Per cent manganese	38.59	51.10	25.81
Per cent manganese by weight	100.00	66.93	33.07
Ratio of concentration	1.987	1.00	

	Fe	Mn	SiO_2	P
Analysis of concentrate, per cent	1.48	51.55	13.14	0.053

Daily crude ore capacity, tons	75
Daily concentrate production, tons	38

Estimated cost of mining crude ore, per ton:	
Mining	$3.00
Tramming and loading	1.25
Royalty	1.50
Unloading	.25
Trucking	2.50
Overhead	1.75
Total	10.25

Value of crude ore, per ton:	
Manganese value, @ $1.02	40.01×$1.02=$40.81
Silica penalty	7× 0.50= 3.50
	5× 0.75= 3.75
	8.91× 1.00= 8.91
Total	16.16
Net f. o. b. Chicago	$40.81− 16.16=24.65
Net value, railroad shipping point, $11.20 freight	24.65− 11.20=13.45
Net profit on sale of crude ore	13.45− 10.25= 3.20

Estimated cost of concentrating plant	$20,000
Estimated amortization period	1 year—300 days
Charge per ton concentrate	$20,000÷(300×38)=$1.75
Interest on investment at 7 per cent per ton of concentrate	(0.07×$20,000)÷11,400=0.12
Estimated cost of concentration per ton	$0.75 per ton of concentrates

Total estimated cost of concentrate on cars:		
Mining	@ $3.00 per ton of crude	1.987×$3.00=$5.96
Tramming	@ .60 per ton of crude	1.987× .60= 1.19
Concentrating	@ .75 per ton of concentrates	1.000× .75= .75
Amortization		1.75
Interest		.12
Royalty at $1.50 per ton of crude		1.987= 1.50× 2.98
Unloading at .25 per ton of concentrate		1.000= .25× .25
Trucking at 2.50 per ton of concentrate		1.000× 2.50= 2.50
Overhead at 1.75 per ton crude		1.987× 1.75= 3.48
Total		18.98

TABLE 9.—*Estimated cost data applied to three California ores*—Continued.

MANGANESE ORE 1009—Continued.

Value of concentrate per ton:	
Manganese value, at $1.24	51.55× $1.24=$63.92
Silica penalty	5.14× .50= 2.57
Net value, f. o. b. Chicago	$63.92− $2.57= 61.35
Net value, railroad shipping point, $11.20 freight	61.35− 11.20= 50.15
Net profit on concentrate per ton	50.15− 18.98= 31.17

Daily profit on crude ore	75×$3.20= $240.00
Daily profit on concentrate	38×31.17=1,184.46
Balance in favor of concentrating	= 944.46

Annual profit on crude ore, 300 days	300×$240.00=$72,000
Annual profit on concentrate, 300 days	300× 944.46×283,338
Balance in favor of concentration	=211,338

Annual production of crude ore, tons	22,500
Annual production of concentrate, tons	11,400

MANGANESE ORE 1011.

	Fe	Mn	SiO_2	P
Analysis of crude ore, per cent	1.27	32.01	39.55	0.029

Concentration results:	Crude ore.	Concentrate.	Tailings.
Per cent by weight	100.00	46.47	53.53
Per cent of manganese	32.19	45.05	21.02
Per cent of manganese by weight	100.00	65.05	34.95
Ratio of concentration	2.152	1.000	

	Fe	Mn	SiO_2	P
Analysis of concentrate, per cent	1.50	45.05	18.64	....

Daily crude ore capacity, tons	75
Daily concentrate production, tons	35

Estimated cost of mining crude ore, per ton:	
Mining	$3.00
Tramming and loading	1.25
Royalty	1.50
Unloading	.25
Trucking	2.50
Overhead	1.75
Total	10.25

Value of crude ore—Too siliceous.

Estimated cost of concentrating plant	$20,000
Estimated amortization period	1 year—300 days
Charge per ton of concentrate	$20,000÷(300×35)=$1.90
Interest on investment at 7 per cent per ton concentrate	(0.07 ×$20,000)÷10,500=.12
Estimated cost of concentration per ton	$0.75 per ton of concentrate.

TABLE 9.—*Estimated cost data applied to three California ores*—Continued.

MANGANESE ORE 1011—Continued.

Total estimated cost of concentrate on cars:		
Mining, at $3 per ton of crude	2. 152×$3. 00=	$6. 46
Tramming, at $0.60 per ton of crude	2. 152× .60=	1. 29
Concentrating, at $0.75 per ton of concentrate	1. 000× .75=	.75
Amortization		1. 90
Interest		.13
Royalty, at $1. 50 per ton of crude	2. 152× 1. 50=	3. 23
Unloading, at $0.25 per ton of concentrate	1. 000× .25=	.25
Trucking, at $2.50 per ton of concentrate	1. 000× 2. 50=	2. 50
Overhead, at $1.90 per ton of crude	2. 152× 1. 90=	4. 09
Total		20. 60
Value of concentrate per ton:		
Manganese value, at $1.12	45. 05×$1. 12=	$50. 46
Silica penalty	7. 0 × 0. 50=	3. 50
Net value	3. 64× 0. 75=	2. 73
Total		6. 23
Net value, f. o. b. Chicago	$50. 46−$6. 23=	44. 23
Net value, railroad shipping point, $11.20 freight	44. 23−11. 20=	33. 03
Net profit on concentrate per ton	33. 03−20. 60=	12. 43
Daily profit on crude ore		
Daily profit on concentrate	35×$12. 43=	$435. 05
Balance in favor		
Annual profit on crude ore, 300 days		
Annual profit on concentrate—300 days	300×$4. 35. 05=	$130, 515
Balance in favor		
Annual production of crude ore, tons		22, 500
Annual production of concentrate, tons		10, 500

MANGANESE ORE 1020.

	Fe	Mn	SiO$_2$	P
Analysis of crude ore, per cent	1. 50	39. 95	25. 10	0. 075

Concentration results:	Crude ore.	Concentrates.	Tailings.
Per cent by weight	100. 00	33. 14	66. 86
Per cent of manganese	38. 77	51. 21	32. 60
Per cent of manganese by weight	100. 00	43. 78	56. 22
Ratio of concentration	3. 018	1. 00	

	Fe	Mn	SiO$_2$	P
Analysis of concentrate, per cent	1. 23	51. 55	11. 65	0. 074

Daily crude ore capacity, tons	75
Daily concentrate production, tons	25

TABLE 9.—*Estimated cost data applied to three California ores*—Continued.

MANGANESE ORE 1020—Continued.

Estimated cost of mining crude ore, per ton:	
Mining	$3.00
Tramming and loading	1.25
Royalty	1.50
Unloading	.25
Trucking	2.50
Overhead	1.75
Total	10.25

Value of crude ore, per ton:		
Manganese value, at $1.00	$39.95×$1.00=	$39.95
Silica penalty	7× .50=	3.50
	5× .75=	3.75
	5.1 × 1.00=	5.10
Total		12.35
Net f. o. b. Chicago	$39.95−$12.35=	27.60
Net value, railroad shipping point, $11.20 freight	27.60− 11.20=	16.40
Net profit on sale of crude ore	16.40− 10.25=	6.15

Estimated cost of concentrating plant		$20,000
Estimated amortization period		1 year−300 days
Charge per ton concentrate	$20,000÷(300×25)=	$2.67
Interest on investment at 7 per cent per ton of concentrate	(0.07×20,000)÷7,500=	0.19
Estimated cost of concentration per ton		$0.75 per ton concentrates

Total estimated cost of concentrate on cars:		
Mining, at $3 per ton of crude	3.018×$3.00=	$9.05
Tramming, at 60 cents per ton of crude	3.018× .60=	1.82
Concentrating, at 75 cents per ton of concentrates	1.000× .75=	.75
Amortization		2.67
Interest		.19
Royalty, at $1.50 per ton of crude	3.018×$1.50=	$4.53
Unloading, at 25 cents per ton of concentrate	1.000× .25=	.25
Trucking, at $2.50 per ton of concentrate	1.000× 2.50=	2.50
Overhead, at $1.75 per ton of crude	3.018× 2.67=	8.06
Total		29.82

Value of concentrate per ton:		
Manganese value, $1.24	51.55× $1.24=	$63.92
Silica penalty	3.65× .50=	1.83
Net value, f. o. b. Chicago	$63.92− 1.83=	62.09
Net value, railroad shipping point, $11.20 freight	62.09− 11.20=	50.89
Net profit on concentrate per ton	50.89− 29.82=	21.07

Daily profit on crude ore	75× $6.15=	$461.25
Daily profit on concentrate	25× 21.07=	526.75
Balance in favor	Concentrating=	65.50

TABLE 9.—*Estimated cost data applied to three California ores*—Continued.

MANGANESE ORE 1020—Continued.

Annual profit on crude ore, 300 days	300×$461.25=	$138,375
Annual profit on concentrate, 300 days	300× 526.75=	158,025
Balance in favor	Concentrating=	19,750
Annual production of crude ore, tons		22,500
Annual production of concentrate, tons		7,500

DISCUSSION OF ESTIMATED COST DATA.

ORE 1009.

The manganese content of this crude ore is within the range of the schedule. The silica content is a few points above, but it has been assumed that a small tonnage of such ore might be accepted. The value of the crude ore has, therefore, been computed.

By a concentration of approximately 2 to 1 a high-grade product can be obtained. Shipment of the crude ore shows a small profit per ton. On concentrate the profit per ton is comparatively large, and, although it weighs much less, the daily profit is shown to be much larger.

ORE 1011.

The manganese content of ore 1011 is below the range of the present schedule and the silica content is very high. It is, therefore, considered that this ore could not be sold as mined, but would require concentration. The concentrate is not very high in manganese, being 5.50 per cent lower than that obtained from ore 1009, although the ratio of concentration is nearly the same. The computation shows a daily profit by concentrating considerably less than for ore 1009.

ORE 1020.

The manganese content of ore 1020 is within the range of the present schedule and the silica content is only 0.1 per cent higher than the rejection limit. Therefore, the value of the crude ore has been computed. The manganese content of the concentrate produced is almost the same as for ore 1009, but the ratio of concentration is much larger. Hence, the profit per ton is less than for ore 1009, although more than for ore 1011. The daily profit is considerably decreased by the smaller output. Consequently, while there is a decided balance in favor of concentrating ore 1009, the balance for ore 1020 is comparatively rather small.

FINENESS OF CONCENTRATES.

The concentration tests of the three ores embodied crushing to pass 10 mesh, classifying, and treating on roughing tables. Ore buyers have said that they were not interested in such fine material.

Indeed, it would not be suitable as a major part of a manganese blast-furnace mixture, but up to 5, or possibly 10 per cent, it should not be excessively harmful. There would be a larger loss in flue dust, and if material of better structure were obtainable it would be greatly preferred by blast-furnace operators. Because of the larger losses attending the use of this fine material, if it were purchased at all, it would probably be penalized.

Possibly such fine concentrates could be agglomerated by briquetting, sintering, or nodulizing. But in practice, the application of any such process to a particular mine is rather doubtful. In the first place, a small daily output would not suffice to keep such a plant in continuous and effective operation. A sintering plant might be erected for the joint use of several mines, or might be erected independently and operated on a custom basis. The first cost of the equipment and the operating costs that would probably prevail in California being high, it is doubtful whether such plants would be profitable, unless there were some assurance of high prices prevailing for a considerable period of time.

At present nodulizing plants at iron blast furnaces in Chicago and elsewhere are running on flue dust. In an emergency they might be turned over to manganese concentrates, and the operating company reimbursed in some manner, possibly by a joint assessment of miner and alloy manufacturer. It is presumed that the sinter would be superior to natural manganese ores as mined and shipped direct, and that a certain part of the cost might be charged off against improved furnace practice.

The time required for the erection of a plant may be considerable, and private capital might not be attracted to an enterprise of this character unless there were some assurance that a given margin of profit might be possible for a period long enough to permit retirement of the original investment for the plant and the gaining of a satisfactory return.

CHAPTER 4.—PREPARATION OF MANGANESE ORE.

By W. R. Crane.

INTRODUCTORY STATEMENT.

The great demand for manganese during the war caused many persons to engage in mining and preparing manganese ores for market. As a result much money was spent on plants that were not adapted to the work. This condition was recognized by the Bureau of Mines early in 1918, and an effort was made to assist the operators of manganese properties by sending engineers into the field to advise as to the proper methods of mining and treating the ores.

A preliminary study of the operations showed that the methods in general use were wasteful; consequently, if practice could be improved it was estimated that the production of the various districts could be increased, by saving a large proportion of the mineral formerly lost—thousands of tons a year. With this object in mind, a standard washing plant was designed, which was based on the best and most successful practice in manganese properties in the country.

The methods employed in cleaning and concentrating manganese ores and the scale of operations depend largely upon the size and character of the deposits worked. The ores obtained from the different forms of deposits are so varied in character that were it economically possible to treat all of them, a wide variation in practice would be necessary. The irregular deposits, consisting largely of manganese and rock fragments inclosed in clay, are probably more largely treated than the regular forms as they produce the bulk of the ores treated. A relatively large amount of ore also comes from the regular or blanket formations, which consist of replacement deposits in the limestone bed rock.

The blanket deposits may be hard ore lying upon or within the limestone bed rock, or may be soft or wad ores. The deposits of wad, some of which are of large size, are usually of low grade and present special problems in washing and concentration; as yet no serious attempt has been made toward their economic treatment.

Washing and concentration methods are discussed herein in the following order: (1) Description of the practice in the various districts and (2) description of a standard plant the design of which is based upon the best practice in the treatment of manganese ore in this country.

The principal difficulty in treating the ore, or wash dirt, from the irregular deposits is the uncertain supply. Next in importance is the character of the manganese mineral and the proportion of ore to waste, such as clay, sand, and iron. The kind of ore best adapted to washing and concentration is the nodular or pebble-like forms of small size; next is the granular form; both are usually of convenient size for log washing and subsequent jig work. The kidney, dornick, and massive ores require reduction before treatment by log washers and on the picking belt, but owing to the size and purity of many of the masses, it is often possible to make a high recovery prior to treatment in logs and jigs.

On account of the uncertain and variable conditions mentioned, the tendency in treating manganese is to reduce the equipment to a minimum, both with regard to kind and number of parts employed. This is frequently done irrespective of the desirability of such limitation. Imperfect cleaning and excessive losses usually result from the curtailment of equipment. However, inadequate equiment of plants is not always due to the wish to limit the first cost, but often to lack of definite knowledge regarding proper methods of treating manganese ores, and the following of local practice, however poor it may be. It is, therefore, important to present details regarding a standard plant that incorporates the most useful and desirable features of the best practice in this country.

METHODS EMPLOYED.

The preparation of manganese ores may be divided into two separate and distinct parts—namely, dry mining, and washing or concentration.

DRY MINING.

Dry mining does not refer to mining, except indirectly, but more to the cleaning of the ore, or wash dirt, by screening it as mined. Not all wash dirt is adapted to dry mining, as some clays are wet and sticky, or occur in large masses, rendering their separation from the manganese mineral by screening practically impossible. Dirt that can be best treated by dry mining is dry and granular, and can be easily broken and separated.

The dry-mining method is unsatisfactory at best and is scarcely ever employed in operations of any size except as a temporary expedient prior to the erection of a washing plant. In small-scale operations where the deposits are small, or sufficient funds to properly equip the mine are lacking, such a method may be permissible. Also, deposits distant from a water supply may require dry mining if they are to be worked at all. In any event the method is wasteful, the loss of fine ore frequently amounting to 25 to 35 per cent of the recoverable mineral in the bank.

On the other hand, the high-grade manganese occurring in solution cavities can be mined and cleaned by hand with or without screening. In such deposits the bulk of the ore is of suitable size and is easily cleaned; however, even under the most favorable conditions, large losses result.

The proportion of manganese in the wash dirt is the controlling factor in dry mining; dirt having less than 10 per cent mineral can not be successfully separated by screening unless the bulk of the mineral is of large size.

WASHING AND CONCENTRATION.

The principal considerations affecting the washing and concentrating of manganese ores are: Quantity of wash dirt available, percentage of manganese in the dirt, and water supply. No plant should be erected until the available supply of ore has been determined within reasonably close limits; failure to test a property properly before erecting a plant shows lack of good business judgment. Ore may occasionally be mined and hand picked under exceptionally favorable conditions of occurrence even when the proportion of ore to clay is 1 to 50, or 2 per cent, but mining must be done on a large scale by machinery and the bulk of the ore must be in fairly large masses. Log washers and jigs can not readily treat materials that are leaner than 1 to 35; the ratio of ore to dirt should preferably be not less than 1 to 10, or 10 per cent, which is a fair average for wash dirt treated.

The design of a plant for the proper treatment of any manganese ore must be based upon the following points: Character of the manganese mineral and its physical properties and occurrence; the associated minerals, and the inclosing rock formations; and the practice previously followed, if any. Of only slightly less importance are the local conditions, such as lay of the ground, water supply, and timber for construction and other purposes.

The character of manganese ore varies widely in different parts of the same district, and even in openings on one property. The mineral may be dense and hard, or porous and hard, or porous and soft. The specific gravity also varies considerably, ranging from 3.5 to nearly 5; the average for the bulk of the mineral treated is between 3.5 and 4. The small rounded or nodular forms and the small granular pieces can be treated to advantage, but fine material and large masses require special treatment.

The principal minerals associated with manganese, as indicated in a previous chapter, are quartz and iron, mostly limonite; barite is of rare occurrence, although occasionally found. As the specific gravity of iron is similar to that of manganese it is practically impossible to effect separation mechanically; consequently it is common practice to hand pick the ore coming from the finishing apparatus when a product low in iron is desired. As a rule, however, no attempt is made to separate the small proportion of iron present in ores from the ordinary manganese deposit.

The specific gravity of quartz is about 2.65, but that occurring in connection with manganese deposits, particularly the sands resulting from the decay of impure limestone and other formations, is often rough and porous and has a specific gravity of 1.8 to 2. It is evident, then, that the average specific gravity of manganese that must be treated is about twice that of quartz, the principal impurity that must be separated. This difference is ample to insure clean separation.

PRACTICE IN CLEANING MANGANESE ORE.

The work of cleaning manganese ores as practiced in the various districts can hardly be considered standard except in general outline—that is, the use of grizzlies, log washers, screens, and picking belts, with a growing tendency toward employing jigs. Rarely are plants constructed fully equipped with the apparatus that have proven useful and satisfactory for treating manganese ores; few plants have a full installation of essential equipment, while a number are operating with the irreducible minimum of equipment, namely, a log washer and a screen, or simply hand jigs.

Practice in the treatment or cleaning of manganese ore varies from fair to bad with a wide range between, but as a whole is poor. Plants are modeled after others already operating, with the assumption, apparently, that because a plant is in operation it must be doing good work, which is often far from being true. However, the essential principles of a prevailing practice are likely to be sound and particularly adapted to the special conditions and needs of the district and are, therefore, worthy of careful consideration and adoption in part at least.

PRACTICE AT A VIRGINIA PLANT.

The Crimora mine is the largest manganese producer in Virginia and until recently has had the reputation of being the largest in the United States. The flow sheet given below shows the method employed in treating a high-grade but rather lean ore.

Flow sheet of Crimora mill, Virginia.

Wash dirt to *1*.

1. Grizzly, 4 by 8 feet, 80-pound rails, spaced 7 inches, to *2*.

2. Revolving screen, 60 inches in diameter by 12 feet long, 3½-inch perforations; oversize—clay to *3*, manganese to *4*; undersize to *5*.

3. Waste bank.

4. Blake crusher, 1½ inches, to *6*.

5. Log washer, double, 25 feet long; slope 1 inch to 1 foot; discharge to *6*; overflow to 7.

6. Revolving screen, 9 feet 8 inches long by 48 inches in diameter, $\frac{1}{4}$-inch perforations; oversize by elevator to *8*; undersize to *9*.
7. Settling pond.
8. Revolving screen, 6 feet long by 40 inches in diameter, 1-inch and $1\frac{1}{4}$-inch perforations: oversize by elevator to *10*: undersize—from 1-inch screen to *11*, from $1\frac{1}{4}$-inch screen to *12*.
9. Revolving screen, 9 feet 8 inches long by 48 inches in diameter, $\frac{1}{4}$-inch perforations; oversize by elevator to *13*; undersize by elevator to *14*.
10. Picking belt, waste, 24 inches wide by 22 feet long; ore to *15*; waste to *16*.
11. Bull jig No. 1, two cells, 36 by 36 inches: gate discharges to *15*; overflow to *16*.
12. Bull jig No. 2, two cells, 36 by 36 inches; gate discharges to *15*; overflow to *16*.
13. Harz jig, three cells, 30 by 40 inches; gate discharges to *17*, hutches to *17*, overflow to 7.
14. Woodbury jig, two cells; gate discharges to *17*; hutches by elevator to *17*; overflow to 7.
15. Picking belt, 30 inches by 22 feet long; ore to *17*, waste to *10*.
16. Waste bin by cars to *3*.
17. Ore bin, finished ore.

PRACTICE AT A PLANT IN GEORGIA.

The flow sheet of the Aubrey plant of the Georgia Iron & Coal Co., given below, represents the best practice and most extensive operation in the Cartersville district, Georgia. The capacity of the plant is 50 tons of finished ore per day of 10 hours. There are three large open cuts that are connected with railroad tracks, so that strippings as well as wash dirt can be handled with equal facility. It is possible that certain parts of the plant might have been arranged to better advantage, yet taken as a whole the design and arrangement are fair.

Flow sheet of the Georgia Iron & Coal Co. plant at Aubrey, Ga.

Wash dirt to *1*.

1. Grizzly (four sets), five railway rails, four spaces of 4 inches each, length 7 feet 3 inches: oversize—rock to *2*; manganese broken by sledge to *3*; undersize to *3*.
2. Rock dump.
3. Log washers (four double logs, 30 feet long); discharge to *4*; overflow to *5*.
4. Screen (perforated metal, openings 1 by 2 inches); oversize to *6*; undersize to 7.
5. Mud or settling pond.
6. Picking belts; rock to *2*; manganese and attached rock to *8*.
7. Belt conveyors to *9*.
8. Crusher reducing to one-half inch to 9.
9. Ore bins by elevator or lift to *10*.
10. Revolving screens for jig feed, Nos. 3, 4, and 5; for jig No. 3, $\frac{3}{16}$-inch, $\frac{1}{4}$-inch, and $\frac{1}{2}$-inch perforations; for jig No. 4, $\frac{3}{16}$-inch, $\frac{1}{4}$-inch, and $\frac{3}{8}$-inch perforations; for jig No. 5, $\frac{1}{4}$-inch, $\frac{3}{8}$-inch, and $\frac{1}{2}$-inch perforations; products from these screens to *11*.
11. Jigs Nos. 3, 4, and 5 (four single cells); gate discharges to *12*; hutches to *13*; overflow to *14*.
12. Concentrates bin to *17*.
13. Flat screen, $\frac{1}{4}$-inch perforations, for jig No. 1; oversize to cell No. 1 of *15*; undersize to cell No. 2 of *15*.

14. Revolving screen, for jig No. 2, ⅛-inch, 3/16-inch, and ½-inch; oversize to cell No. 1, undersize to cells Nos. 2, 3, and 4, No. 2 jig.
15. Jig No. 1 (two cells); gate discharges to *12;* hutches to *12;* overflow to *5.*
16. Jig No. 2 (four cells); gate discharges to *12;* hutches to *12;* overflow to *5.*
17. Hand-picking table, where ore is sorted by attendants, who pick out iron and other impurities. The iron is sold as manganiferous iron ore.

PRACTICE IN THE BATESVILLE DISTRICT, ARKANSAS.

There were 11 washers built in the Batesville district, Arkansas, during 1917 and 1918. None of these did satisfactory work, largely because the builders and operators did not appreciate the necessity of providing screens for properly sizing the material treated on the jigs.

The practice in this district is simple and may be outlined as follows:

Flow sheet used in Batesville district, Arkansas.

Wash dirt to *1.*
1. Grizzly, size 5 by 6 feet, 10 to 14 railroad rails spaced 4 inches; oversize—waste rock thrown on dump, lump ore reduced to size that will pass through grizzly to *2;* undersize to *2.*
2. Double log washer, 28 feet long, slope 1 inch to 1 foot; discharge to *3;* overflow to *4.*
3. Revolving screen, 6 feet long by 30 inches in diameter, with ¾-inch perforations; oversize to *5;* undersize to *6.*
4. Waste bank.
5. Picking belt, 22 feet long by 24 inches wide; ore to *7;* waste to *4.*
6. Harz jig (four cells); size of cells 22 inches by 42 inches; gate discharges to *7;* hutches to *8.*
7. Finished ore bin.
8. Harz jig (two cells), size 28 inches by 36 inches; gate discharges to 7, hutches to 7, tails to *4.*

Rarely are more than two jigs employed in the plants for washing manganese ores, and in many plants only one jig is used. Revolving screens or trommels are commonly used, but flat screens are occasionally employed. In either case it is the exception rather than the rule to find more than one size of opening, which ranges from ¾ inch to 1¼ inches. The wide range in size fed to the jigs from such screens renders good separation next to impossible. Few of the jig products are clean, and they require extensive hand picking to complete the preparation. This hand sorting could be obviated by simply rerunning the hutch products of the rougher jigs; fairly clean products could then be made on the second or cleaner jigs. When only one jig is employed, few or none of the jig products is suitable for market. Fairly close sizing would greatly improve the work done by the jigs and reduce the expense of preparation by eliminating hand picking.

Probably a dozen hand jigs have been in use in the district at different times. As with the power jigs, incomplete separation of waste was the almost universal rule, little or no attention being paid to sizing of the feed.

PRACTICE AT A MILL AT PHILIPSBURG, MONT.

The mill of the Philipsburg Mining Co. is one of the largest in the West, having a capacity of 75 to 100 tons per day. The ores treated are the oxides mined at Philipsburg, Mont. The flow sheet of the mill follows:

Flow sheet of Philipsburg Mining Co. at Philipsburg, Mont.

Wash dirt (ore) to *1*.

1. Grizzly, 2-inch spaces; oversize to *2*, undersize to *3*.
2. Gyratory crusher, No. 4, 1¼ inches, to *3*.
3. Ore bin to *4*.
4. Revolving screen, 1¼-mm. mesh; oversize to *5*, undersize by elevator No. 1 to *6*.
5. Picking belt; clean ore to *7*, rejected by elevator No. 1 to *6*.
6. Revolving screen, two sections—first, ⅜-inch mesh, second, ¾-inch mesh; oversize of first to second, undersize of first to *8*, oversize of second to *6*, undersize of second to *9*.
7. Clean-ore bin.
8. Revolving screen, 4-mm. mesh; oversize to *10;* undersize to *11*.
9. Bull jig; concentrates to *12;* overflow to *13*.
10. Middling jig; concentrates to *12;* overflow to *13*.
11. Revolving screen, 1½-mm. mesh; oversize to *14;* undersize to *15*.
12. Dorr classifier; spigot to *16;* overflow to *17*.
13. Rolls, ⅛-inch space, by elevator No. 2 to *18*.
14. Fine jig; concentrates to *12*, overflow to *13*.
15. Dewatering tank; spigot to *19*, overflow to *17*.
16. Frue vanner; concentrates by No. 3 elevator to *20*, overflow to *17*.
17. Settling pond.
18. Revolving screen; 2½-mm. mesh, oversize to two coarse jigs, undersize to *11*.
19. Distributor to *21*.
20. Drier to *22*.
21. Wilfley tables; concentrates to *16*, tailings to *23*.
22. Concentrates bin.
23. Waste bank.

DESCRIPTION OF A STANDARD WASHING PLANT.

The cleaning of manganese ores by washing and jigging can readily be accomplished and that, too, with slight change in the present practice. The flow sheet of a plant properly arranged to do satisfactory work is given below.

Flow sheet of standard washing plant.

Wash dirt to *1*.

1. Grizzly, 2-inch to 4-inch spaces between bars; oversize rock to *2*; manganese by sledge to *3*; undersize to *3*.
2. Rock dump.
3. Log washer, double log, 20 to 30 feet long; discharge to *4*; overflow to *5*.
4. Revolving screen, cylindrical or conical, perforations or mesh ½-inch and 3/16-inch; oversize, everything above ½-inch, to *6*; undersize, ½-inch to 3/16-inch, to 7; 3/16-inch and smaller, to *8*.
5. Mud or settling pond.
6. Picking belt; rock to *2*; manganese to *9*.

7. Rougher jig, four cells; gate discharges to *9*; first hutch to *9*; second, third, and fourth hutches to *10*.
8. Sand jig, three cells; gate discharges to *9*; hutches to *9* or *10*.
9. Finished ore bin.
10. Cleaner jig, three cells; gate discharges to *9*; hutches to *9* or *11*.
11. Shaking table; finished product to *9*; tailing to *5*.

A standard washing plant having the above flow sheet may be called a single washer or unit, and has a capacity of 40 to 50 tons of finished ore per 10 hours. An increase in capacity of such a plant, aside from a small increase gained through crowding, would mean doubling, trebling, or quadrupling of most of the equipment listed, with corresponding increase in the volume of wash dirt treated.

By single washer is not meant a single log, but a single washer with double logs. Single logs do good work, yet in a plant as outlined, a double log should be employed.

The sizes of the various apparatus of a standard plant are indicated in the flow sheet; the sizes of screens and adjustments of the various apparatus for proper operation are given below.

LOGS.

The number of revolutions of logs varies from 12 to 15 per minute. The amount of water required varies widely with the nature of the bank dirt, but is usually 50 to 75 gallons per minute. The capacity likewise varies with the bank dirt; average dirt should yield 40 to 50 tons product per 10 hours. Roughly, 25 horsepower is required to drive logs.

SCREENS.

The number of revolutions is 15 to 20 per minute, being approximately the same for cylindrical and conical screens 36 to 48 inches in diameter. The power required for driving varies somewhat with feed and size of material handled, but is about one horsepower. The capacity of screens of the sizes given ranges from 45 to 55 tons per 10 hours for the smaller, and 50 to 75 tons for the larger. For the ordinary material treated the screen should be not less than 60 nor more than 72 inches long. If a larger number of sizes is desired, it might be better to shorten the length of the separate screening surface than to increase unduly the length of the whole screen.

PICKING BELTS.

The sizes of picking belts commonly employed, which have proved satisfactory for the usual range of work done, are: 30 feet long by 18 inches wide to 50 feet long by 30 inches wide. The speed of the picking belt ranges from 50 to 60 linear feet per minute. The capacity is limited by the character of the work; if the material sorted is largely ore and the waste rock is fairly coarse, the capacity is large, similarly where the percentage of ore is small and the individual pieces are large; but where the ore and waste are about equal in amount and there is a wide range in sizes, the capacity may be small. Occasionally two picking belts are used, one for coarse, the other for the finer materials, but such practice is neither necessary nor desirable in a standard plant.

JIGS.

The following data show the proper adjustment and operation of jigs:

Data regarding size and operation of jig equipment.

(Dimensions in inches.)

	Width.	Length.
Size of sieve compartments	20 to 24	30 to 38
Size of plunger compartments	18 to 22	30 to 38

	1st cell.	2nd cell.	3rd cell.	4th cell.
Size of sieve openings:				
Rougher jig	$\frac{1}{4}$	$\frac{1}{4}$	$\frac{3}{16}$	$\frac{3}{16}$
Sand jig	$\frac{3}{16}$	$\frac{3}{16}$	$\frac{1}{8}$	$\frac{1}{8}$
Cleaner jig	$\frac{3}{16}$	$\frac{3}{16}$	$\frac{1}{8}$	$\frac{1}{8}$
Length of stroke of jigs:				
Rougher jig	$1\frac{1}{2}$	$1\frac{1}{2}$	$1\frac{1}{4}$	$1\frac{1}{4}$
Sand jig	1	1	$\frac{7}{8}$	$\frac{7}{8}$
Cleaner jig	1	1	$\frac{7}{8}$	$\frac{7}{8}$
Number of strokes per minute:				
Rougher jig	150	150	150	150
Sand jig	200	200	200	200
Cleaner jig	200	200	200	200

Power for 4-cell jig, 2.5 to 3.0 hp.

Power for 3-cell jig, 2.0 to 2.5 hp.

Hydraulic water pipes: Main pipe 2 inches; branch pipes 1 inch.

Amount of water used by a jig, 350 to 400 gallons per minute.

Height of tailboard or dams, $3\frac{1}{2}$ to $4\frac{1}{2}$ inches.

Slope of bed (sieve) should not exceed 1 inch to length of bed; with heavy mineral, bed should be level. Slope should be against, not with movement of mineral.

Drop between tailboards or dams, 1 inch.

Height of gate discharge above bed, 2 to $2\frac{1}{2}$ inches. This must be varied to suit conditions and character of material treated.

Jigs to have full or large capacity must be fed regularly; therefore jigs receiving the feed directly or indirectly from logs usually do unsatisfactory work for the reason that irregular feeding does not permit the maintenance of a bed of uniform depth. Further, the feeding of materials which vary from sandy to clayey in character also renders it difficult to keep a full bed; clayey ores tend to clog the jig beds and are preferably cleaned by trommels and jets of water.

With a suitably arranged plant, practically all materials treated may be handled by gravity, but in places elevators are required. Elevators, although adding to the equipment and the expense of operation, permit a better arrangement, in that more of the apparatus may be on the same level, and therefore more accessible.

Elevators are particularly useful for raising the hutch products from the rougher and sand jigs to the cleaner jigs. Should it be found desirable to employ a shaking table in reworking the hutch products from the cleaner jigs, elevators should also be employed in handling them.

TABLES.

It is doubtful whether tables can be used to advantage in the preparation of the usual run of manganese ore, as outlined and discussed. However, there is a well-defined field for their application in treating certain ores, such as the low-grade blanket deposits and breccia ores.

DRY CONCENTRATION.

Largely through the lack of an adequate supply of water a number of attempts have been made to clean manganese ores by dry methods. Probably the most extensive and elaborate equipment is that of the Southern Hill Manganese Co. on the Southern Hill property, in the Batesville district, Arkansas.

It is proposed to separate the ore from the clay and rock fragments by passing the material as excavated through a revolving drum, where it will be thoroughly dried and the clay pulverized. On being discharged from the drum the fine material will be removed by a revolving screen, which is a continuation of the drum. The coarse manganese and waste will then pass to a picking belt, where final separation is to be made. The smaller sizes of ore are to be treated on hand jigs. This plant has never been operated on a commercial scale; consequently no definite statement can be made regarding its practicability.

The water supply for manganese washing and concentrating plants is important, as many of the workable properties are often at considerable distance from an available supply. Pipe lines 3 to 4 inches in diameter are sufficiently large for a standard plant. The conservation of water by passing it through settling or impounding ponds usually solves the problem of water supply where the quantity of fresh water is limited. Once a plant has been put in operation the addition of 25 per cent of the total consumption not only provides for the loss due to wastage, but furnishes the required amount of fresh water for those steps of the process requiring clean water.

GENERAL SUMMARY OF CONDITIONS AFFECTING CONCENTRATION.

On account of the irregularity of manganese deposits and the uncertainty of an adequate supply of ore, great care should be exercised in connection with the various operations, particularly with respect to prospecting and mining. Similarly, preparatory to the erection of a washing or concentrating plant, the factors that have to do with the success of the work must be carefully considered.

The principal considerations affecting the cleaning of ores are:

1. The character and grade of ore.
2. The recoverable percentage of mineral.
3. Relative value of crude to cleaned ore.
4. Basis upon which royalty is paid, whether crude ore or the cleaned product.

Aside from the clay and other materials more or less intimately mixed with manganese considerable silica is associated with the ores. The silica may be "free" or "attached." The free silica can be readily removed by washing, but the attached silica, being embedded in the ore or attached to it can be separated, if at all, with difficulty.

High-grade ores, particularly when occurring in large masses, and soft ore, as pyrolusite, should receive the minimum preparation consistent with proper cleaning. Low-grade ores usually require much more careful treatment than the high-grade ores, and the work and expense of concentration depend largely upon the impurities present. Free silica is not difficult to separate from the manganese. Soft ore, or wad, although of high grade, is difficult to clean without great loss from fines, particularly when much fine sand is mixed with the clays and ore.

The recoverable percentage of mineral in the wash dirt depends largely upon the character of the mineral. Certain clays are readily broken and separated from the manganese, whereas others become pasty when washed, adhering tenaciously to the particles of mineral. As a rule, the larger the pieces of ore and the higher the grade the more readily is separation from the waste effected, owing probably to the smoother surfaces. Nodular ore of small and fairly uniform size is readily washed and jigged, but fragments from large masses and rough particles resulting from decay of limestone and possible partial solution of manganese are very difficult to clean satisfactorily.

The relative value of crude ore as compared with that of cleaned ore may be the deciding factor in determining whether a concentrating plant should be erected. During the past year considerable quantity of low-grade ores was shipped at a low price, simply because there was a market for it. The question is whether such ores could not have been raised in grade by concentration, so as to have brought a price that would have warranted the erection of a suitable plant. However, uncertainty as to extent of the deposits and the length of time the prevailing schedule of prices would be maintained did not foster experiments of this sort.

The grade of ore upon which royalties are assessed has been the cause of considerable trouble in different districts, but in most of the districts during the past year royalties were paid on all ores coming within the schedule unless otherwise specified. In the future high-grade ores will alone be subject to royalty charges.

In future careful mining in well-proved deposits of high-grade ore will be necessary in the various manganese districts. The ores mined will, in turn, require either close hand-picking or concentration in well-designed plants in order to produce a high-grade ore, low in silica and phosphorus. With a dependable output of such ore, it should be possible to continue operation in the face of foreign competition wherever freight rates to consuming furnaces are reasonably favorable.

CHAPTER 5.—LEACHING OF MANGANESE ORES WITH SULPHUR DIOXIDE.

By C. E. van Barneveld.

INTRODUCTORY STATEMENT.

Early in 1918 the Bureau of Mines, through its mining experiment station at Tucson, Ariz., began an investigation of possible methods of recovering manganese from the numerous deposits of low-grade, siliceous, secondary manganese ores that are scattered throughout Arizona, New Mexico, and Utah. The aggregate quantity of such ore is large, but as a rule these ores are not amenable to ordinary methods of concentration, and the proportion of ore that can be brought to shipping grade by sorting and screening is usually small. Much of this ore is found in localities tributary to the copper-smelting districts where waste sulphurous roaster gases are available. The possibility of using sulphur dioxide as a leaching agent for manganese had long been recognized, and it was thought that a process developed at the Tucson station for leaching certain copper ores with hot sulphurous fumes might be successfully applied to this type of manganiferous material. In May, 1918, some leaching tests were made at this station on manganese ores from the Patagonia district in Arizona. The results were sufficiently encouraging to lead to an extensive program of research into the possibility of developing an SO_2-leaching process which would produce a high-grade sinter containing over 60 per cent metallic manganese in the form of oxides, free from silica and phosphorus. The investigation had not been completed when the economic situation as regards manganese was entirely changed by the armistice and further experimentation was abandoned. However, sufficient work had been done to warrant certain conclusions. As these may have a scientific interest and a bearing upon other chemical and metallurgical operations, the following brief report is presented.

The work was undertaken with due recognition of certain facts:

1. That prompt action was essential.

2. That the process would probably have to be applied at smelting centers where sulphurous fumes were available as waste products, and that the cost of transportation of ores to the smelter would be a limiting factor in its application.

3. That a plant of 250-ton capacity would be the desirable unit and that in all probability two such units would be the maximum installation for any one smelter.

4. That from 4 to 6 months would be required for construction and that the cost of the plant would have to be written off over an operating period of 12 or even 6 months.

5. That the cooperation of the copper-smelting companies in Arizona and Utah and of the large consumers of manganese would be necessary in order to arrange for prompt plant construction; also that it might be possible and necessary to use public funds.

The investigation was carried through the laboratory stage on a sufficiently large scale to warrant as a wartime measure the construction on the results obtained of a commercial unit. On October 1 negotiations were under way for the erection of a 250-ton unit at one of the copper smelters, in cooperation with a large producer of lead-zinc-silver-iron-manganese ore in which only the fluxing value of the manganese was being realized. While this negotiation was in progress the rapid improvement in the military situation so changed the outlook that the undertaking was abandoned before final plans were drawn.

RESULTS OF TESTS.

The tests made included ores from the Clifton, Bisbee, and Patagonia districts in Arizona; also representative ores from New Mexico, Utah, Nevada, and California. The results of the tests are presented in Table 10 following:

TABLE 10.—*Results of tests of manganese ores by sulphur dioxide leaching with continuous-drum process.*

Source of ore.	Mn in heads.	Weight of ore taken for test.	Size of ore (mesh).	Pulp ratio.	Temperature of solution.	Rate of feed per hour.	Mn in tails.	Percentage of Mn recovered.	Remarks.
	Per cent.	*Pounds.*			° *C.*	*Pounds.*	*Per cent.*		
Parker, Ariz.	11.9	500	20	2-1	50	200	1.25	89.5	Highly siliceous ore containing 58 per cent SiO_2, 4 per cent Fe, 5 per cent CaO.
Bisbee, Ariz.	15.4	200	20	2-1	51	100	4.2	73	High lime ore (33 per cent CaO).
Do	15.6	200	30	2-1	52	100	6.3	58	High iron ore (32 per cent Fe).
Do	15.2	200	20	2-1	50	100	2.56	83	General ore.
Do	7.12	100	20	4-1	47	100	1.85	75	Highly siliceous ore (59 per cent SiO_2).
Tombstone, Ariz.	16.6	102	20	4-1	47	50	.99	93.9	Silver-copper ore.
Do	20.1	101	20	4-1	47	50	.85	95	Do.
McCarty, Delta, Utah	30.4	100	20	4-1	45	50	3.4	90	Contained 20 per cent SiO_2, 7 per cent Fe, 16 per cent CaO.
Hardshell, Patagonia	26.3	250	30	2-1	50	200	4.2	84	Silver ore (10 ounces Ag per ton).
Prince consolidated ore, Pioche, Nev.	11.5	100	20	2-1	50	100	.6	91.8	Lead-silver-gold ore containing 33 per cent Fe, 4 per cent CaO, 11 per cent SiO_2.
Wad ore, California	33.	....	20	4-1	50	....	2.3	93	Sample by C. E. van Barneveld.
Montgomery, Nev	13.3	100	20	4-1	46	....	5.5	59	High lime ore (31 per cent CaO), containing considerable manganese as carbonate and silicate.

OBSERVATIONS ON RESULTS OF TESTS.

The following observations, based entirely on work done with dilute hot sulphurous gases (under 6 per cent SO_2) used countercurrent to the flow of the pulp, are presented:

THE ORE.

The general run of western ores may be described as impure psilomelane resulting from alteration of silicate or carbonate. The impurities may include numerous substances such as lime, silica (which may or may not be combined), and iron in the form of limonite or siderite.

Such an ore is readily leached with SO_2 and need rarely be crushed finer than 20 mesh to insure a good extraction.

THE PULP.

The pulp should be broken into a fine spray to insure contact between the mineral particles and the reagent. A leaching drum was designed at this station for the lixiviation of nonsulphide copper ores which gives the necessary agitation and contact.

A patent (U. S. patent No. 243015) was applied for and was granted on November 8, 1918.

Briefly, this apparatus comprises a drum set horizontally, to rotate slowly. The interior is divided into compartments by a series of transverse partitions, each compartment having longitudinal baffles and peripheral lifters. The transverse partitions are perforated so that the pulp may pass through the successive compartments to the end or discharge compartment. The pulp is fed into the first compartment and drops to the bottom of the compartment; as the drum rotates the peripheral lifts raise the pulp until it is spilled onto the horizontal staggered baffles below in such manner that the pulp is splashed and distributed over the surfaces of the bars and against the transverse partitions in fine descending drops and particles, thereby insuring intimate and prolonged contact between the ore particles and the countercurrent sulphurous and oxidizing gases. With a constant feed a flow is established within the drum, so that the pulp is gradually passed through the successive compartments into the last or discharging compartment, the lifters of which raise the pulp and drop it into a discharge or exit pipe whereby it is conducted through a trap opening into a discharge launder.

The pulp enters one end of the drum cold, and the SO_2-charged air or gas enters the opposite end hot. The countercurrent flow developed results in the pulp becoming progressively warmer and the gas becoming correspondingly cooler, until the pulp at the discharge end is heated to any desired temperature. When SO_2 is introduced as a hot dilute gas containing less than 6 per cent SO_2 by volume (the balance of the gas being largely air, which has lost part of its oxygen and which may contain the various impurities commonly found in roaster gases) and is projected into a fine spray of hot pulp or solution, absorption of SO_2 is practically negligible and the hot pulp or solution will contain practically no free SO_2 and will discharge from the drum in a practically neutral condition.

SOLUBILITY OF THE MANGANESE MINERALS.

The various higher manganese oxides, especially pyrolusite, psilomelane, and wad, were found to be readily soluble in the hot sulphurous acid formed by introducing hot air or furnace gas containing 2 to 6 per cent SO_2 (by volume) countercurrent to a pulp flow having a consistency of 2 of water to 1 of ore crushed to 20 mesh. Manganite and braunite are not commercially soluble.

Silicates of manganese are insoluble in sulphurous acid at atmospheric pressure and only slightly soluble under pressure.

Carbonates of manganese may, from an operating standpoint, be considered insoluble. The reagent slowly attacks and decomposes the carbonate, but some of the dissolved manganese at once reprecipitates as a sulphite; this precipitate coats the remaining undissolved carbonate and effectually prevents further dissolution.

IRON.

The effect of SO_2 on iron minerals differs markedly from its effect on manganese minerals. Iron in the oxide form, such as magnetite, hematite, and limonite, is practically insoluble in SO_2. Metallic iron and iron in the form of carbonate (siderite) is readily soluble. This iron will be in solution as a ferrous sulphite or sulphate; it will, however, precipitate as a basic ferric compound in the presence of either $CaCO_3$ and oxygen or $ZnCO_3$ and oxygen; it will also be similarly precipitated from solution so long as there is undissolved MnO_2 in the ore charge.

PHOSPHORUS.

In some forms phosphorous dissolves readily in SO_2. No large scale tests in the drum were made on high phosphorus ores. Laboratory tests, however, indicate that by the use of tandem drums phosphorus may be eliminated in the following manner: The phosphorus will not be dissolved in the first drum (the discharge from which still contains much undissolved manganese and other bases). Whatever dissolved phosphorus is discharged from the second drum will return to the ball mill. The addition of large quantities of fresh ore will at once neutralize these solutions, and the phosphorus will thereupon precipitate. If sufficient soluble iron and phosphorus are present to cause the building up of these elements by resolution in the drum after precipitation in the ball mill, then the filtrate from the second drum may be passed over any available carbonates, such as $CaCO_3$, prior to being returned to the ball mill. The iron and phosphorus in solution would then be precipitated and eliminated from the circuit.

LIME.

In the presence of an appreciable amount of MnO_2 the concentration of free SO_2 is insufficient to decompose $CaCO_3$. As dissolution of MnO_2 progresses, some $CaCO_3$ will be dissolved and precipitated as $CaSO_4$ (insoluble). The catalytic action of the iron and manganese oxides will convert a small proportion of the entering SO_2 to SO_3; the resulting sulphuric acid will convert $CaCO_3$ to $CaSO_4$. Some of the $CaCO_3$ particles become coated with gypsum and are not further acted upon by the acids in the drum.

ZINC AND COPPER.

Zinc and copper in nonsulphide form are readily dissolved by SO_2, but sulphides of these metals are not attacked.

TREATMENT OF THE PREGNANT SOLUTION.

The discharge from the drum will consist of a pulp containing in solution as sulphates the manganese, zinc, copper, and perhaps some iron; as solids the gangue, the sulphides, the other insoluble minerals, and any dissolved lime reprecipitated as $CaSO_4$.

The pregnant solution may be separated from the solids by filtration and washing in pressure filters.

Any copper present in the filtrate may be recovered by precipitation on iron.

The manganese-zinc content is removed together, either by crystallization, in large-scale permanent installations, where the lowest operating costs are to be sought, or by evaporation in pans with artificial heat in smaller or in temporary installations, where minimum installation expense would regulate the choice.

The resulting sulphate is roasted for two hours, more or less, in a rotary type of clinker furnace at a temperature ranging from 825° to 1,000° C.

The product from the roaster is a hard, compact clinker which in the zinc-free ores will run from 60 to 64 per cent manganese. Any remaining manganese sulphate may be removed by water leaching; this leaching does not seem to cause the cake to crumble. The waste heat from this operation is used to heat the cast-iron evaporation pans. The SO_2 gas may be used over again as a solvent if desired.

The manganese sulphate commences to decompose around 700° C. and should break up completely at 830° C. It was found that in order to effect complete decomposition of the sulphate with reasonable quickness (say in 30 minutes) the roast should be preformed at considerably higher temperature—around 950° C. Plenty of oxygen is necessary. In a reducing atmosphere some manganese sulphide will be formed; once formed, this can only be broken up by first reoxidizing to the sulphate.

CONDITIONS ESSENTIAL IN COMMERCIAL PRACTICE.

Three essential conditions must be complied with in commercial operations.

(*a*) In order that dissolution may be effected rapidly a reasonable strength of gas is required—not less than 2 per cent and up to 6 per cent of SO_2—the balance of the gas being heated furnace products.

(*b*) Heat aids dissolution. It is desirable to have an admission temperature of gas which will give the discharging solutions a temperature of 40° C. minimum to 60° C. maximum.

(*c*) The time of contact is regulated by the rate of flow of pulp through the drum. The efficiency of extraction is directly proportionate to the pulp surface exposed to the gas. This in turn depends upon proper agitation and breaking up of the pulp into a very fine spray which must be so directed countercurrent to the gas that no particle of mineral can escape contact with the reagent. This can be satisfactorily accomplished in 15 to 30 minutes in the drum described.

LEACHING IN TWO STAGES.

Manganese sulphate solutions are near the point of saturation when they contain 11 per cent manganese (specific gravity 1.5). As the solution approaches the point of saturation the rate of dissolution decreases markedly. On the other hand, maximum concentration of solutions minimizes the expense of subsequent evaporation to dryness, which is the next step in the process. Therefore, leaching should be performed in two stages in two tandem drums as indicated on the flow sheet shown in figure 1.

Wet crushing in ball mills is advised in circuit with drag classifiers, the filtrate from the second drum discharge being used to pulp the ore.

The discharge from the first drum will be a practically neutral pulp made up of partly leached ore, which will still contain considerable undissolved manganese, and a solution containing $MnSO_4$ near the point of saturation, and also $ZnSO_4$ and $CuSO_4$ if nonsulphide zinc and copper are present in the ore. This solution will not contain iron. This pulp is then filtered on a pressure type of filter and the various products are disposed of as follows:

(*a*) The filtrate is treated for recovery of valuable metals as previously described.

(*b*) The filter cake and wash water are repulped in a pulping tank, together with the wash water from the second drum filtration, to form the feed for the second drum, where the remaining SO_2-soluble minerals, including perhaps some iron, are dissolved.

The discharge from the second drum is filtered and the various products are disposed of as follows:

(*a*) The residue or filter cake goes to waste, or to a briquetting plant in case the lead-silver or copper-silver or other values warrant further treatment such as smelting.

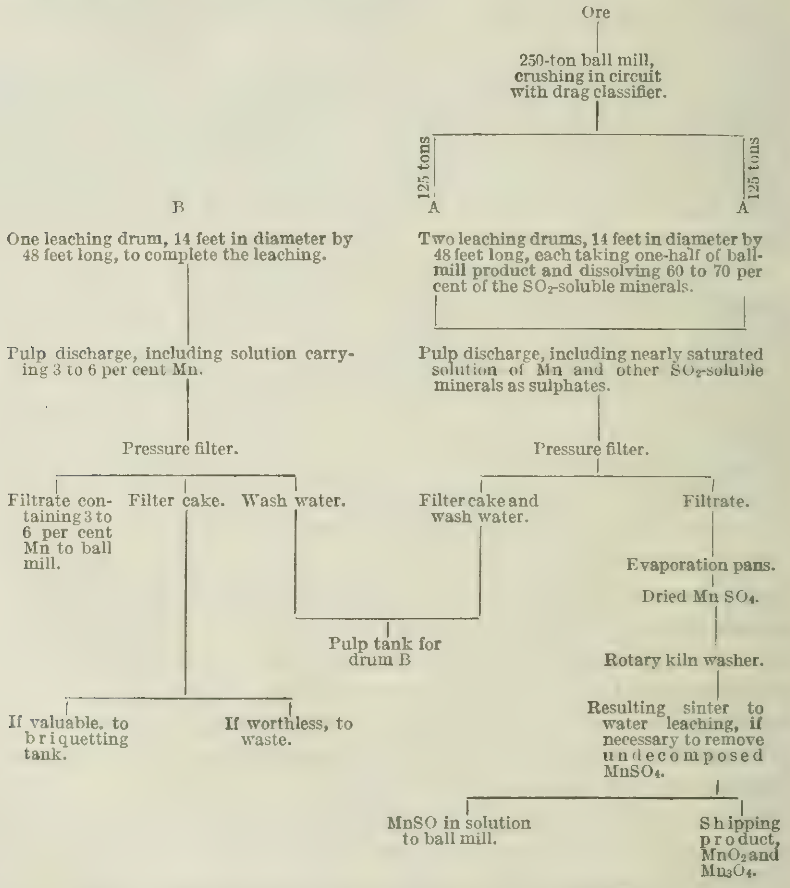
Ore
250-ton ball mill, crushing in circuit with drag classifier.
125 tons
125 tons
A
A
B
One leaching drum, 14 feet in diameter by 48 feet long, to complete the leaching.
Two leaching drums, 14 feet in diameter by 48 feet long, each taking one-half of ball-mill product and dissolving 60 to 70 per cent of the SO2-soluble minerals.
Pulp discharge, including solution carrying 3 to 6 per cent Mn.
Pulp discharge, including nearly saturated solution of Mn and other SO2-soluble minerals as sulphates.
Pressure filter.
Pressure filter.
Filtrate containing 3 to 6 per cent Mn to ball mill.
Filter cake.
Wash water.
Filter cake and wash water.
Filtrate.
Evaporation pans.
Dried Mn SO4.
Pulp tank for drum B
Rotary kiln washer.
If valuable, to briquetting tank.
If worthless, to waste.
Resulting sinter to water leaching, if necessary to remove undecomposed MnSO4.
MnSO in solution to ball mill.
Shipping product, MnO2 and Mn3O4.

FIGURE 1.—Flow sheet of proposed plant for SO_2 leaching of manganese ores.

(*b*) The filtrate, which may contain up to 5 or 6 per cent manganese, goes to the ball mill to pulp the original ore for the first drum.

(*c*) The washwater from filtration goes to the pulping tank in which the filtercake from the first filtration is pulped prior to being fed into the second drum.

The crushing, leaching, filtration, and washing are thus carried on in a circuit. Any iron dissolved in the second drum is returned to the ball mill and is there immediately precipitated.

FILTERS.

A pulp made of 20-mesh ore and a saturated, slimy $MnSO_4$ solution presents an interesting filtration problem. . Tests on a semicommercial scale proved that the vacuum type of filter could be used only by the addition of filter cells and by dilution of the $MnSO_4$ solution considerably below the saturation point. In a series of tests made under the direction of A. W. Hudson, of the Phelps Dodge Corporation, on manganiferous silver ore from Tombstone, Ariz., vacuum filters proved inadequate. The addition of filter cells partly nullified the concentration of silver values obtained by extraction of the manganese. The dilution of the pregnant liquid appreciably below the point of saturation increased the cost of subsequent evaporation of $MnSO_4$.

A test made on a pressure type of filter gave the following results: The quantity of pulp passed through the filter contained 3,200 pounds of solid matter and yielded 4,000 pounds of cake containing 20 per cent water. The filter yielded 6.6 pounds of filter cake (20 per cent water) per square foot per hour. On this basis a plant to treat 250 tons of dry ore would yield 312.5 tons of filter cake (20 per cent water) and would require $(312.5 \times 2{,}000) \div (6.6 \times 24) = 3{,}946$ square feet of filtering surface.

CORROSION.

Corrosion of metal in contact with the solution was found by experiment to be due entirely to the action of ferrous sulphate and liberated sulphuric acid. Leaching in two stages presents a simple solution of this difficulty. As sulphur dioxide has a marked affinity for manganese, conditions in the first drum may be so regulated as to avoid the dissolution of any iron, and the resulting pregnant liquid will not corrode the evaporation pans. The solution from the second drum may be neutralized and the dissolved iron may be precipitated over carbonates prior to the return of this solution into the crushing end of the circuit, or this may be done in the ball mill.

SULPHUR DIOXIDE.

Sulphur dioxide for leaching purposes may be obtained (*a*) by burning flower sulphur in sulphur burners; (*b*) by roasting high-grade iron pyrites containing sufficient copper or silver to give the calcines a market value; (*c*) as a waste product from roasting furnaces at a copper smelting plant.

Approximately three-fourths of a unit of sulphur is required on an average for each unit of manganese, iron, zinc, and lime dissolved. Most of the sulphur dioxide consumed in leaching is recovered in the final roasting of the sulphate and can be reused. Sulphur in combination with lime is lost. In view of the plant requirement, the relatively small size of the average western manganese deposit, and the uncertainty of the manganese market over a period of years, it would seem that with sulphur dioxide reduction of straight manganese ores must be confined to favorably situated copper smelting plants where sulphur dioxide of the proper strength is available as a waste product.

COMPLEX ORES.

The foregoing observations relate largely to ores in which manganese is the principal constituent. Some study was also given to the more complex manganiferous ores, such as silver ores, lead-silver ores, copper-silver ores, and zinc ores, containing sufficient manganese and occurring in sufficiently large quantities to warrant their consideration as a possible emergency source of manganese. Much laboratory work was done in an attempt to determine the relative solubility in sulphur dioxide of the various minerals that may be present in a complex ore, in the hope that some selective action might be discovered which would effect a separation between manganese and zinc. However, it was impossible to reproduce or to maintain in the drum the conditions that in the laboratory might produce certain preferential action on oxides over carbonates. Under operating conditions a large excess of acid is always present in some part of the drum; and the result is that the various SO_2 soluble manganese, zinc, and copper minerals are attacked practically simultaneously. If anything, zinc minerals (with the exception of crystalline zincite) will be dissolved fastest; the higher manganese oxides are next in the scale and the copper minerals last.

As the three minerals soluble in SO_2, copper, zinc, and manganese, are recovered together as sulphates in the filtrate, their separate recovery from sulphate solution was investigated. Copper offers no difficulty; it may be recovered by passing the solution over iron. In order to retain the manganese in solution during the precipitation of the copper, the specific gravity of the pregnant solutions in treating copper-bearing ores, must be lessened. The copper in solution will be replaced with iron which will, of course, result in a slightly lower manganese content in the sinter. Some preliminary work was done on the separation of zinc and manganese, which may be briefly summarized as follows:

(*a*) A straight oxidizing roast around 1,050° C. converts the sulphates to oxides. If these oxides are mixed with powdered coal and are then heated in a retort to 1,500° C., the zinc volatilizes with a recovery of 80 per cent of condensed metallic zinc.

(*b*) A 60 per cent zinc recovery as oxide is possible by subjecting the sulphates to a straight oxidizing roast until converted to oxides and then adding coal or other reducing agent and continuing the roast in the presence of air at 1,350° C.

(*c*) A chloridizing roast will cause the zinc to volatilize as chloride. If sodium chloride is used the soda fuses with the mass, thus preventing complete chloridization. If calcium chloride is used there is no fusion. The lime is converted to sulphate (insoluble in SO_2), part of the manganese becomes oxidized and part remains as sulphate. The latter is water-soluble and the oxidized manganese can be extracted by redissolving it with SO_2, an operation which calls for too much manipulation and expense to be commercially practicable except under very favorable conditions.

(*d*) Electrolysis does not seem practicable. The zinc goes to the cathode and the manganese to the anode. The H_2SO_4 set free at both poles attacks the zinc. Apart from this, there are the troubles engendered by foul solutions.

This investigation is incomplete and should receive further attention.

COSTS.

GENERAL ESTIMATE.

The cost of a 250-ton unit erected at the Salt Lake smelters or at the Douglas, Arizona smelter, would in normal times be roughly $100,000. A 500-ton unit should not cost to exceed $150,000. Under present (1918) conditions a 250-ton unit would cost perhaps as much as $200,000. The operating cost would be $12 to $14 per ton of shipping product based on an 80 per cent recovery of manganese from an ore which has a manganese content of 20 per cent. This includes crushing, leaching, and filtering 4 tons of ore; drying the filtrate and roasting the dried manganese sulphate. The resulting shipping product would be a sinter, free from silica, alumina, and phosphorus, which in the ores free from nonsulphide zinc or copper would contain 60 to 64 per cent manganese as oxides.

The cost of SO_2 treatment per ton of ore at a smelter where waste SO_2 gases are available should fall well within the estimated cost of $3.50 indicated in the following table.

Estimate cost of treating one ton of manganese ore.

Crushing in gyratory and ball mill	$0.40
Disposal of tailings	.10
Operation of drums and filtration plants	.15
Evaporation of $MnSO_4$ and roasting (heat, labor, and supplies)	[a] 1.50
Incidental handling	.25
Power	.30
Water	.10
General overhead	.20
Charge for SO_2	.50
	3.50

---

[a] This item is high.

CALCULATED ESTIMATES FOR TYPICAL ORES.

The following estimates cover typical examples in Utah and Arizona. A railroad freight rate of $12 per long ton on shipment of sinter to eastern points has been assumed.

EXAMPLE 1.

Assume that a large siliceous ore body is located 10 miles from the main line of a railroad. The manganese content varies from 15 to 40 per cent, and the silica, 30 to 20 per cent, the balance being alumina, lime, and a little iron, with no zinc or copper. Average manganese content of the run-of-mine ore is 20 per cent, and all the manganese is soluble in SO_2.

Four tons of ore will (on an 80 per cent recovery basis) produce 1 short ton of high-grade sinter containing 60 per cent manganese, no silica, alumina, or phosphorus. The price, according to May, 1918, schedule is $1.30 per unit plus $6.50 silica premium = $84.50 per long ton, f. o. b. Chicago, or $84.50 − $12 freight = $72.50 per long ton f. o. b. smelters. The costs will be as follows:

Market value of one short ton shipping product, f. o. b. smelters, based on above schedule		$64.74
Cost of mining 1 ton of ore	$2.00	
Haulage to railroad at 30 cents per ton-mile	3.00	
Loading, sampling, switching and general handling	1.00	
Railroad freight to smelter	2.50	
SO_2 treatment charge	3.50	
SO_2 plant extinguishment	2.00	
Cost producing and treating 1 ton of ore	14.00	
Cost of producing 1 short ton of shipping product		56.00
Net profit on 4 tons of ore		8.74
Net profit per ton of ore mined		2.18

It is assumed that the market will hold for 12 months after completion of the SO_2 plant, or that 12 months' production has been contracted for, and that the entire cost of the plant (at war-time cost of $200,000) must be charged off against 100,000 tons of ore. The plant-extinguishment charge will therefore, be $2 per ton, as given in the table.

EXAMPLE 2.

Assuming that the shipping grade of the ore in example 1 could readily be raised to 25 per cent manganese at an additional mining cost of 50 cents per ton and to 30 per cent manganese at an additional cost of $1 per ton, the returns would be, respectively, as follows:

Four tons of 25 per cent ore will yield 1.19 long tons of 60 per cent manganese sinter, having a market value of [($84.50 − $12 freight)] × 1.19 = $86.30. The profit under these conditions would be [($86.30 − ($56 + 2)] ÷ 4 = $7.08 per ton of sorted ore at the mine.

Four tons of 30 per cent ore will yield 1.4 long tons of 60 per cent manganese sinter, having a market value of ($84.50 − $12 freight) × 1.4 = $101.50 f. o. b. smelter. The profit would be [($101.50 − ($56 + 4)] ÷ 4 = $10.37 per ton of sorted ore at the mine.

Evidently both these sorted ores would stand a larger haulage charge.

EXAMPLE 3.

Assume that a manganiferous lead-silver ore has a lime-silica gangue with sufficient iron and lime to balance the silica after removal of the manganese. The manganese content is 20 per cent. This has a fluxing value of $1.40. The removal of the manganese effects a shrinkage of smelter tonnage, which is in turn offset by a briquetting charge on the lixiviation tailings. The lead-silver contents are assumed to have a net value of $4 per ton of original ore if the manganese is removed and $6.40 per ton of original ore if smelted without removal of manganese.

Assuming the same general conditions and prices as obtained in example 1, it is evident that this would be a highly profitable wartime operation, as the $4 smelter return per ton of original ore would be added to the $2.18 profit on the manganese content per ton of ore mined, making a total profit of $6.18 per ton of ore mined. An otherwise unprofitable operation would thus be made highly profitable and an important source of manganese would be developed.

EXAMPLE 4.

Assuming that under normal economic conditions there would be a ready eastern market for a 60 per cent manganese sinter at 75 cents per unit, the value of the manganese f. o. b. smelter would shrink to about $28 and the maximum return per ton of original ore would be $28 + $16 = $44.

Under normal conditions there would be an appreciable sealing down of operating costs and this operation would show a small profit as follows:

Market value at smelter of the manganese and other metals in four tons of ore		$44.00
Mining per ton of ore	$1.50	
Haulage to railroad at 20 cents per ton-mile	2.00	
Loading, sampling, and general handling	1.00	
Railroad freight to smelter	2.50	
SO_2 treatment charge	3.00	
SO_2 plant extinguishment and interest on investment (based on 4 years)	.50	
Cost of producing and treating 1 ton of ore	10.50	
Cost of mining and treating 4 tons of ore necessary to produce 1 short ton 60 per cent manganese shipping product		42.00
Profit on 4 tons or ore at mine		2.00
Profit on 1 ton of ore at mine		.50

A slight increase in either the manganese content or in the lead-silver content on the one hand or a decrease in transportation costs on the other hand would show a satisfactory balance.

POSSIBLE FUTURE APPLICATION OF SO_2 METHOD.

The war-need incentive to further investigation of this question by the Bureau of Mines no longer exists. It is probable, however, that there are certain properties where the foregoing information might be profitably applied, as these figures offer some encouragement when considered in connection with fairly large ore bodies which, in addition to the manganese, may contain other valuable minerals, such as (*a*) 1 to 2 per cent of copper (whether sulphide or nonsulphide); (*b*) sufficient lead or silver, or combined lead and silver to leave a residue that would pay to smelt; (*c*) sufficient zinc to warrant separating the manganese and the zinc at the initial treatment plant or to warrant experimentation along the lines of electric smelting and subsequent recovery of the zinc from the slag.

CHAPTER 6.—THE JONES PROCESS FOR CONCENTRATING MANGANESE ORES; RESULTS OF LABORATORY INVESTIGATIONS.[a]

By PETER CHRISTIANSON and W. H. HUNTER.

INTRODUCTORY STATEMENT.

During the summer of 1918 a "direct-reduction" process for treating manganese ores, known as the Jones process from the name of its inventor, John T. Jones, was investigated at the laboratory of the Minneapolis experiment station of the Bureau of Mines. This work was undertaken because the process offered possibilities for concentrating the manganese in the manganiferous iron ores of the Cuyana Range and elsewhere.

The importance of this work in relation to war needs has passed, but a record of the investigation may be of general interest to the industry.

The aim of the Jones process is to effect by metallurgical means a separation and consequent concentration of the manganese in ores in which the manganese is so intimately associated with the iron that ordinary methods of gravity or magnetic separation are impracticable.

The process consists of two stages. In the first, called the low-temperature reduction stage, two products are made, namely, (*a*) metallic iron, suitable for direct use in the manufacture of steel, and (*b*) slag. Most of the manganese is concentrated in the slag. Separation of these two products, as outlined by the inventor, is accomplished by grinding the sinter and passing it over a magnetic separation apparatus, producing thereby a high-grade magnetic iron concentrate, and a nonmetallic manganiferous sinter. This method was used in the preliminary tests. In the final test, however, separation was accomplished by pouring the material under treatment in a liquid condition and recovering the iron metal in the form of a button. This last method of separation, developed at the Minneapolis station of the bureau, has obvious advantages, and should the process become a commercial success, would doubtless be the method used.

The second stage, which may be called the high-temperature stage, involves smelting the manganiferous slag or sinter derived from the first stage, to produce a manganese alloy.

---

[a] Prepared in cooperation with the school of mines of the University of Minnesota.

LOW-TEMPERATURE REDUCTION.

PRELIMINARY TESTS.

OUTLINE.

The ore used in preliminary tests was typical of those locally known as disseminated Cuyuna ores, containing approximately 24 per cent iron, 18 per cent manganese, and 26 per cent silica. The following temperatures were tried: 1050°, 1150°, 1250°, 1300°, and 1350° C.

The time was varied from one-half hour to three hours.

The reducing agent used was Elkhorn coal, containing 6.6 per cent ash and 40.8 per cent volatile matter; the amount was varied from 10 to 50 per cent of the ore.

APPARATUS USED.

The apparatus used in the tests were as follows:

Dixon graphite crucibles, size No. 2 for 200 grams of ore, and others in proportion to size of charge.

Gas muffle furnaces for temperatures of 1050° and 1150° C.; oil-fired muffles for 1250° C.; electric carbon resistor furnace for 1300° and 1350° C.

Pyrometers: Hoskins base-metal couples and Hoskins meters for temperatures up to and including 1250° C.

Platinum-and-platinum-rhodium couples and Leeds & Northrup potentiometer for temperatures above 1250° C.

Hand-crushing apparatus, screens, hand magnets, and panning apparatus for separating magnetic from nonmagnetic material.

The temperatures indicated are those of the muffle outside the crucibles. The temperatures of the charges within the crucibles were probably nearly the same, except as regards those charges heated only 30 minutes. Temperatures of heats above 1250° C. were taken within the charge. Readings were generally taken every five minutes.

PROCEDURE.

CHARGING AND HEATING.

When muffle furnaces were used the ore and the reducing agent, crushed to pass 10 mesh, were mixed and charged into cold crucibles; these were placed in the muffle heated to nearly the desired temperature. When crucible furnaces were used the charge was put into crucibles heated to nearly the desired temperature. By using the muffle furnaces a number of crucibles could simultaneously be subjected to practically the same temperatures, thus keeping this variable constant; and a variation in time could be obtained by removing the crucibles at the end of different periods of time.

DISCHARGING.

At the end of the reduction period the contents of the crucibles were discharged into cast-iron molds, so as to effect a rapid cooling.

SEPARATION.

After cooling, the content of each crucible was generally separated into the following:

$$\left.\begin{array}{l}\text{1. Metal}+30\text{ mesh}\\ \text{2. Metal}-30\text{ mesh}\end{array}\right\}\text{magnetic part.}$$

$$\left.\begin{array}{l}\text{3. Slag}\\ \text{4. Carbon.}\end{array}\right\}\text{nonmagnetic part.}$$

This separation was effected by a series of crushings, screenings, and magnetic separations, frequently supplemented by panning. Each of these parts, if panned, was weighed and sampled for analysis.

ANALYSES.

The determinations generally made were: Per cent soluble iron; per cent total iron; per cent soluble manganese; per cent total manganese; per cent insoluble.

CALCULATIONS.

The results obtained from the analyses, together with the weights of the various parts, were combined by calculations to obtain the metal content and percentages of iron and manganese recovered in each of total magnetic and nonmagnetic parts.

The following formulas were used in calculating:

Formulas used in calculating recoveries.

Percentage Extraction.

For magnetic part.

Grams Fe.

Grams of magnetic part+30×per cent Fe content=........................ ....

Grams of magnetic part−30×per cent Fe content=........................ ....

Sum=total grams Fe in magnetic part=........................ ....

$$\frac{\text{Grams Fe in magnetic part}\times 100}{\text{Grams Fe in charge}}=\text{per cent extraction of Fe.}$$

For nonmagnetic part.

Grams Fe.

Grams of slag×per cent of Fe content=........................ ....

Grams of carbon×per cent of Fe content=........................ ....

Sum=total grams Fe in nonmagnetic part=........................ ....

$$\frac{\text{Grams Fe in nonmagnetic part}}{\text{Grams Fe in charge}}\times 100=\text{per cent extraction Fe.}$$

Recovery Per 100 Units of Ore.

Per cent Fe or Mn in ore×per cent extraction=Recovery per 100 units of ore.

Per Cent Soluble Iron in Concentrates.

Grams soluble Fe.

Per cent soluble Fe×grams of magnetic part+30=........................ ....

Per cent soluble Fe×grams magnetic part−30=........................ ....

Sum=Total grams soluble Fe in magnetic part=........................ ....

$$\frac{\text{Sum}\times 100}{\text{Grams of magnetic part}}=\text{per cent soluble Fe in magnetic part.}$$

Per Cent Soluble Iron in Soluble Part of Magnetic Part.	Grams soluble matter.
Per cent soluble matter×magnetic+30=...............................	....
Per cent soluble matter×magnetic−30=...............................	....
Sum=Total grams soluble matter in magnetic part=................	....

$$\frac{\text{Total grams soluble Fe in magnetic part}}{\text{Grams soluble matter in magnetic part}}\times 100=\text{per cent soluble Fe in soluble part of magnetic part.}$$

Results obtained in the tests are given in Table 11.

TABLE 11.—*Results of preliminary low-temperature reduction tests with the Jones process.*

RESULTS WITH VARYING TEMPERATURES.

(*a*) DATA.

Test No.	16–D.	17–D.	35–D.	31.	32.
Temperature, °C.	1,050	1,150	1,250	1,300	1,350
Ore, grams	363	363	160	363	363
Coal, grams	91	91	40	91	91
Time, minutes	120	120	120	120	120

(*b*) EXTRACTION.

Temperature °C.	Product.	Weight, grams.	Iron.		Manganese.	
			Total, per cent.	Recovery, per cent.	Total, per cent.	Recoverery, per cent.
	Ore	363.0	24.36	....	18.12	....
1,050	Magnetic part	233.5	32.0	84.7	19.8	70.5
1,050	Nonmagnetic part	84.6	5.7	5.5	19.3	24.8
1,150	Magnetic part	131.4	58.5	86.9	15.0	30.0
1,150	Nonmagnetic part	189.4	8.18	17.5	20.9	60.2
[a] 1,250	Magnetic part	40.5	90.6	93.9	3.2	4.5
[a] 1,250	Nonmagnetic part	85.3	2.0	4.4	28.9	84.8
1,300	Magnetic part	94.8	87.2	93.5	7.0	10.0
1,300	Nonmagnetic part	209.0	2.4	5.7	26.2	83.0
1,350	Magnetic part	96.6	80.4	91.0	9.4	14.3
1,350	Nonmagnetic part	202.2	4.4	10.0	23.7	72.8

[a] In test 35–D, 160 grams of ore was used.

(*c*) RECOVERY.

Test No.	Variable temperature, °C.	Iron.			Manganese.		
		Grams in 100 grams of ore.	Recovery.		Grams in 100 grams of ore.	Recovery.	
			Per cent.	Grams.		Per cent.	Grams.
16–D	1,050	24.36	84.7	20.5	18.12	70.5	12.60
17–D	1,150	24.36	86.9	21.1	18.12	30.0	5.44
35–D	1,250	24.36	93.9	22.7	18.12	4.5	0.80
31	1,300	24.36	93.5	22.6	18.12	5.7	1.03
32	1,350	24.36	91.0	22.0	18.12	14.3	2.60

RESULTS WITH VARIATION OF TIME, AT 1,150° C.

(*a*) DATA.

Test No.	17–A.	17–B.	17–C.	17–D.	17–E.
Time, minutes	30	60	90	120	150
Ore, grams	363	363	363	363	363
Coal, grams	91	91	91	91	91
Temperature, °C.	1,150	1,150	1,150	1,150	1,150

RESULTS WITH VARIATION ON TIME, AT 1,150° C.—Continued.

(*b*) EXTRACTION.

Minutes.	Product.	Weight, grams.	Iron.		Manganese.	
			Total, per cent.	Recovery, per cent.	Total, per cent.	Recovery, per cent.
	Ore	363.0	24.36	....	18.12	....
30	Magnetic part	156.0	38.7	68.4	15.8	37.5
30	Nonmagnetic part	163.0	10.0	18.6	18.8	34.8
60	Magnetic part	204.4	38.4	89.0	17.7	55.5
60	Nonmagnetic part	125.5	7.1	10.1	21.3	40.6
90	Magnetic part	166.0	44.8	84.3	18.7	47.0
90	Nonmagnetic part	154.2	8.4	14.6	22.7	53.3
120	Magnetic part	131.4	58.5	86.9	15.0	30.0
120	Nonmagnetic part	189.4	8.18	17.5	20.9	60.2
150	Magnetic part	122.7	57.0	79.3	13.6	25.2
150	Nonmagnetic part	178.5	8.1	16.4	22.1	59.9

(*c*) RECOVERY.

Test No.	Variable, minutes.	Iron.			Manganese.		
		Grams per 100 grams of ore.	Recovery.		Grams per 100 grams of ore.	Recovery.	
			Per cent.	Grams.		Per cent.	Grams.
17-A	30	24.36	68.4	16.6	18.12	37.5	6.8
17-B	60	24.36	89.0	21.7	18.12	55.5	10.1
17-C	90	24.36	84.3	20.5	18.12	47.0	8.5
17-D	120	24.36	86.9	21.2	18.12	30.0	5.4
17-E	150	24.36	79.3	19.1	18.12	25.2	4.6

(*d*) SOLUBLE IRON IN MAGNETIC PART.

Minutes.	Product.		Weight, grams.	Total, grams.	Percentage of soluble iron	Grams of soluble iron.	Total grams of soluble iron.	Percentage of soluble iron.
30	Magnetic part	+30	....	....	....	....	....	....
		−30	....	....	....	....	....	....
60	Magnetic part	+30	131.5	204.4	46.0	60.5	77.2	37.7
		−30	72.9		22.9	16.7		
90	Magnetic part	+30	135.1	166.0	47.8	64.7	72.6	43.6
		−30	30.9		25.4	7.9		
120	Magnetic part	+30	84.15	131.4	62.8	52.8	68.4	52.0
		−30	47.2		33.0	15.6		
150	Skull	..	50.9	122.7	44.0	22.4	66.6	54.4
	Powder	..	71.8		61.6	44.2		

(*e*) SOLUBLE IRON IN SOLUBLE PART OF MAGNETIC PART.

Minutes.	Product.		Weight, grams.	Total grams.	Percentage of soluble matter.	Grams of soluble matter.	Total grams of soluble matter.	Total grams of soluble iron.	Percentage of soluble iron in soluble matter.
30	Magnetic part	+30	....	....	....	....	....	....	....
		−30	....	....	....	....	....	....	....
60	Magnetic part	+30	131.5	204.4	75.5	98.8	144.4	77.2	53.4
		−30	72.9		62.2	45.6			
90	Magnetic part	+30	135.1	166.0	73.0	98.8	117.4	72.6	61.5
		−30	30.9		59.3	18.6			
120	Magnetic part	+30	84.15	131.4	81.7	68.8	99.1	68.4	69.7
		−30	47.2		64.2	30.3			
150	Skull	..	50.9	122.7	67.2	34.2	91.0	66.6	73.4
	Powder	..	71.8		79.2	56.8			

RESULTS WITH VARIATION OF TIME, AT 1,250° C.

(a) DATA.

Test No.	35–A.	35–B.	35–C.	35–D.	35–E.
Time, minutes	30	60	90	120	150
Ore, grams	160	160	160	160	160
Coal, grams	40	40	40	40	40
Temperature, °C	1250	1250	1250	1250	1250

(b) EXTRACTION.

Minutes.	Product.	Weight, grams.	Iron.		Manganese.	
			Total per cent.	Recovery per cent.	Total per cent.	Recovery per cent.
	Ore	160.0	24.36	....	18.12	....
30	Magnetic part	67.4	38.26	66.2	16.7	38.6
30	Nonmagnetic part	78.7	10.8	21.8	20.5	55.5
60	Magnetic part	44.0	79.2	89.2	4.8	7.3
60	Nonmagnetic part	92.0	3.3	7.7	27.4	86.9
90	Magnetic part	39.7	88.6	90.4	4.3	5.9
90	Nonmagnetic part	93.8	4.9	11.8	26.3	85.2
120	Magnetic part	40.5	90.6	93.9	3.2	4.5
120	Nonmagnetic part	85.3	2.0	4.4	28.9	84.8
150	Magnetic part	42.7	91.0	98.5	4.0	5.9
150	Nonmagnetic part	81.7	1.6	3.3	29.0	81.7

(c) RECOVERY.

Variable, minutes.	Iron.			Manganese.		
	Grams per 100 grams of ore.	Recovery.		Grams per 100 grams of ore.	Recovery.	
		Per cent.	Grams.		Per cent.	Grams.
30	24.36	66.2	16.1	18.12	38.6	7.1
60	24.36	89.2	21.7	18.12	7.3	1.3
90	24.36	90.4	22.0	18.12	5.9	1.1
120	24.36	93.9	22.8	18.12	4.5	0.8
150	24.36	99.5	24.2	18.12	5.9	1.1

(d) SOLUBLE IRON IN MAGNETIC PART.

Minutes.	Products.	Weight, grams.	Total, grams.	Percentage of soluble iron.	Grams of soluble iron.	Total grams of soluble iron.	Percentage of soluble iron.
30	Magnetic part +30	....	....	....	....	....	....
	Magnetic part −30	....	....	....	....		
60	Magnetic part +30	34.3	44.0	81.0	27.8	34.8	79.0
	Magnetic part −30	9.7		71.8	7.0		
90	Magnetic part +30	33.7	39.7	90.8	30.6	35.2	88.6
	Magnetic part −30	6.0		77.0	4.0		
120	Magnetic part +30	36.4	40.5	92.3	33.6	36.6	90.5
	Magnetic part −30	4.1		71.95	3.0		
150	Skull ..	39.1	47.7	91.5	35.8	38.4	9.00
	Powder ..	3.6		71.75	2.6		

Results with Variation of Time, at 1,250° C.—Continued.

(c) SOLUBLE IRON IN SOLUBLE PART OF MAGNETIC PART.

Minutes.	Product.	Weight, grams.	Total, grams.	Percentage of soluble matter.	Grams of soluble matter.	Total grams of soluble matter.	Total grams of soluble iron.	Percentage of soluble iron in soluble matter.
30	Magnetic part +30	....	....	....	....	....	....	....
	Magnetic part −30	....		....	....			
60	Magnetic part +30	34.3	44.0	90.0	30.9	39.4	34.8	88.5
	Magnetic part −30	9.7		88.0	8.5			
90	Magnetic part +30	33.7	39.7	98.2	33.1	38.5	35.2	91.5
	Magnetic part −30	6.0		90.0	5.4			
120	Magnetic part +30	36.4	40.5	100.0	36.4	40.0	36.6	91.2
	Magnetic part −30	4.1		87.8	3.6			
150	Magnetic part +30	39.1	42.7	99.6	39.0	42.2	38.4	91.0
	Magnetic part −30	3.6		87.7	3.2			

Results With Variation of Amount of Reducing Agents.

(a) DATA.

Test No.	34-A.	34-B.	34-C.	34-D.
Coal, grams	20	40	60	80
Ore, grams	200	200	200	200
Temperature, °C	1,250	1,250	1,250	1,250
Magnetic part, grams	64.6	53.5	50.1	52.7
Nonmagnetic part, grams	94.2	109.7	121.6	127.6

(b) EXTRACTION.

Coal, per cent of ore.	Product.	Weight, grams.	Iron.		Manganese.	
			Total, per cent.	Recovery, per cent.	Total, per cent.	Recovery, per cent.
	Ore	200.0	24.36	....	18.12	....
10	Magnetic part	64.6	68.3	90.5	10.7	19.0
10	Nonmagnetic part	94.2	6.1	11.7	27.4	71.3
20	Magnetic part	53.5	90.0	98.8	4.7	13.0
20	Nonmagnetic part	109.7	3.6	7.4	28.3	85.8
30	Magnetic part	50.1	89.9	92.5	4.8	6.5
30	Nonmagnetic part	121.6	4.1	10.2	26.0	87.2
40	Magnetic part	52.7	70.5	76.2	13.5	19.6
40	Nonmagnetic part	127.6	10.7	27.9	21.1	74.3

(c) RECOVERY.

Coal, percentage (by weight) of ore.	Iron.			Manganese.		
	Pounds per 100 pounds of ore.	Recovery.		Pounds per 100 pounds of ore.	Recovery.	
		Per cent.	Pounds.		Per cent.	Pounds.
10	24.36	90.5	22.0	18.12	19.0	3.5
20	24.36	98.8	24.0	18.12	7.2	1.3
30	24.36	92.5	22.5	18.12	6.5	1.2
40	24.36	76.2	18.5	18.12	19.6	3.6

The foregoing results have been plotted in figures 2 to 9.

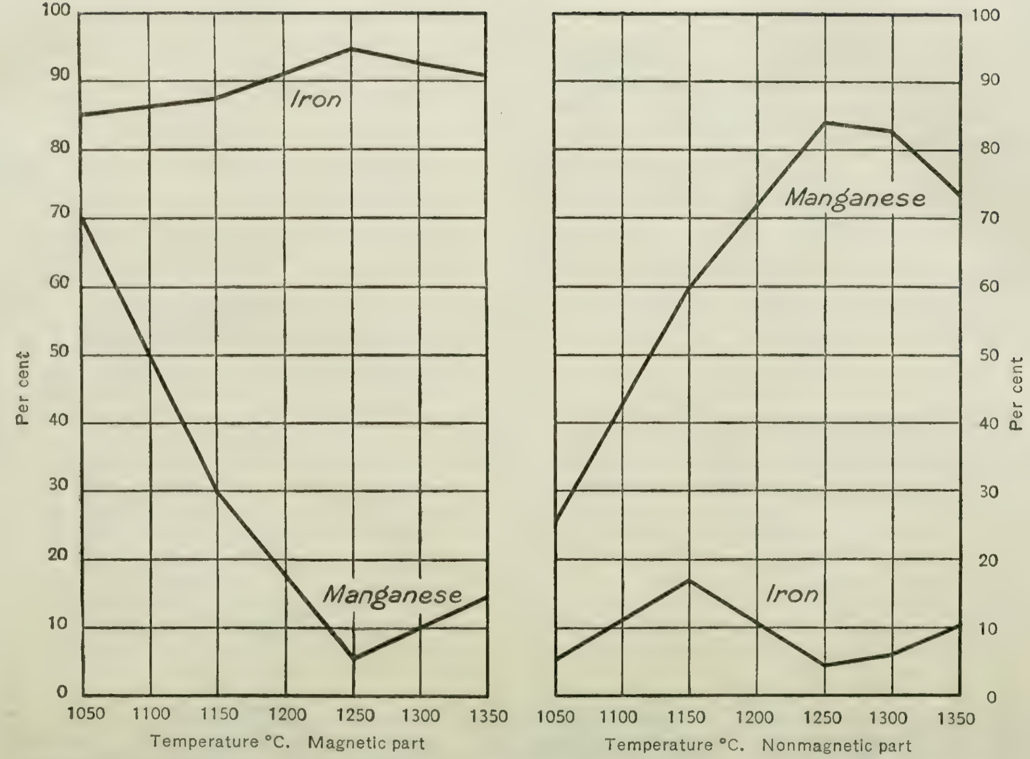

FIGURE 2.—Curves showing recoveries of iron and of manganese in heats 16, 17, 32, and 35, with varying temperatures.

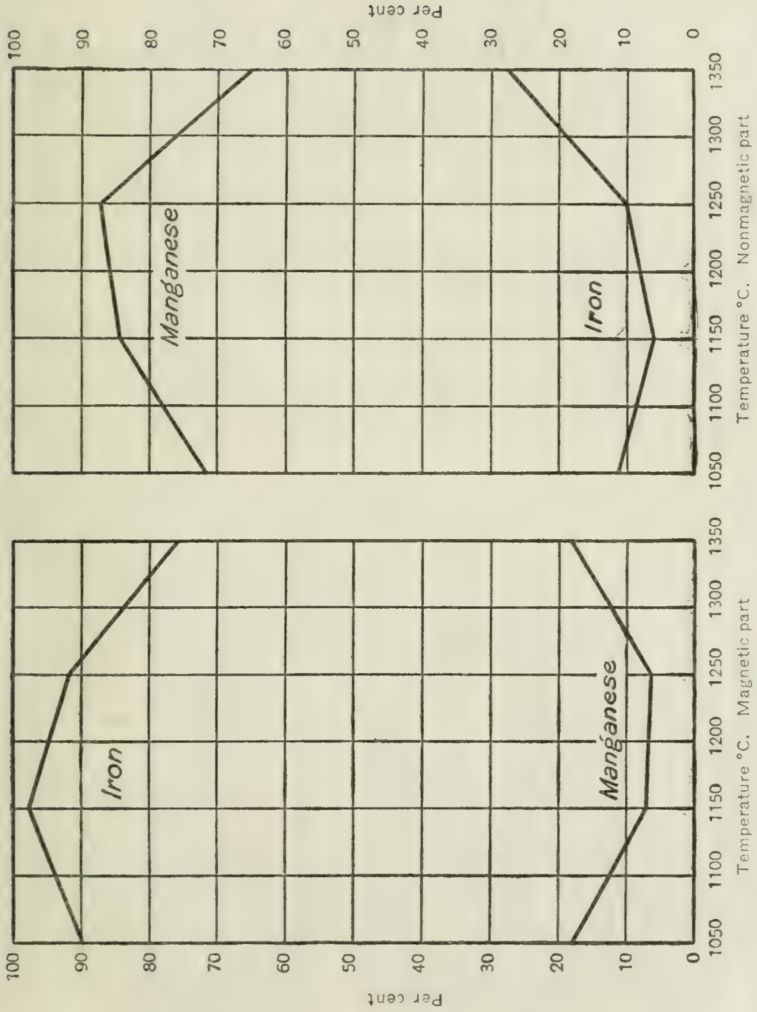

FIGURE 3.—Curves showing recoveries of iron and of manganese in heat 34 at 1,250° C. and with varying quantities of reducing agent.

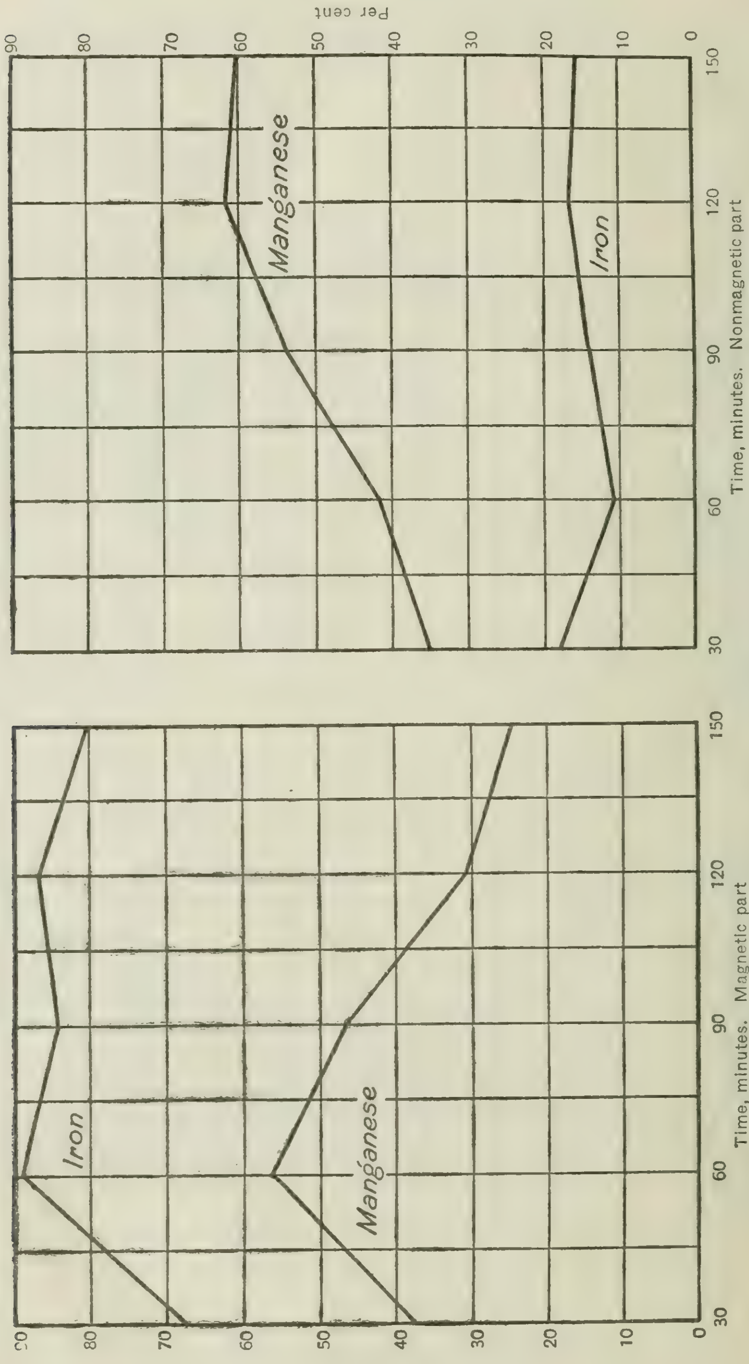

FIGURE 4.—Curves showing recoveries of iron and of manganese in heat 17 at 1,150° C. and with varying time.

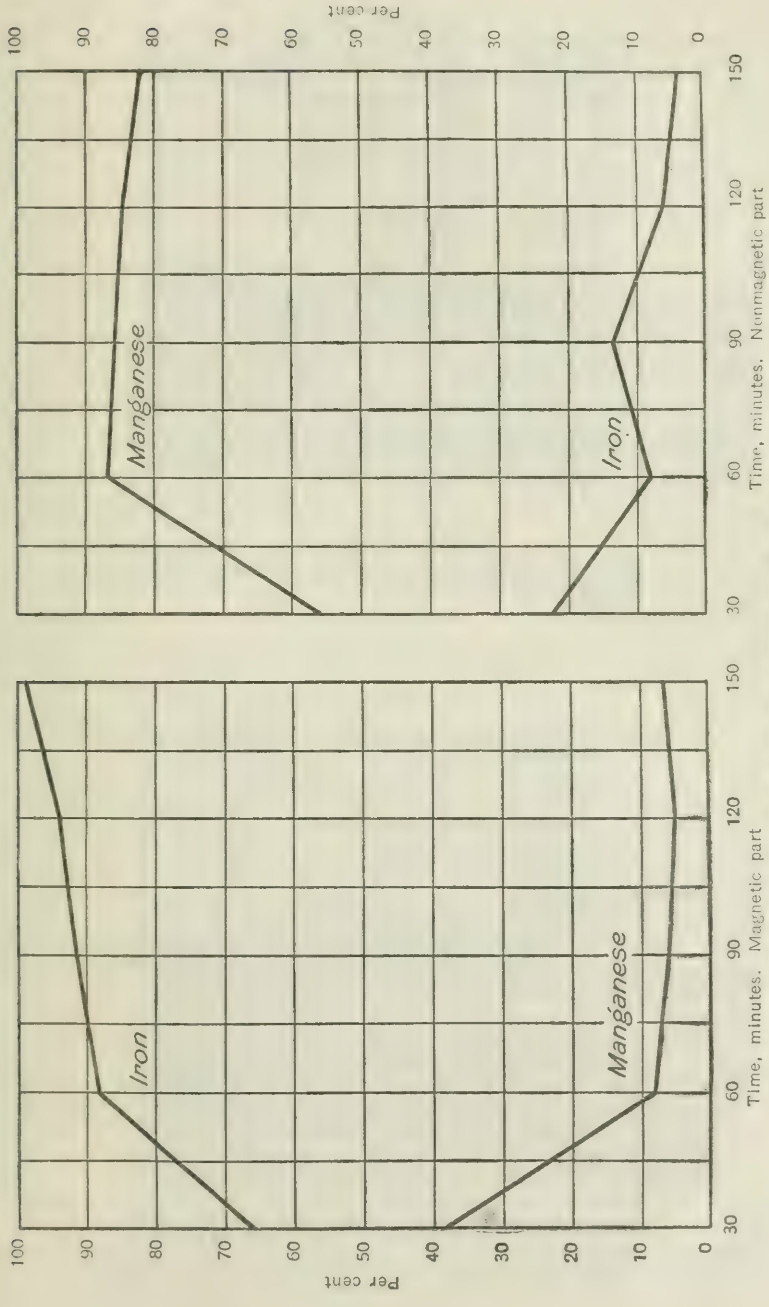

FIGURE 5.—Curves showing recoveries of iron and of manganese in heat 35 at 1,250° C. and with varying time.

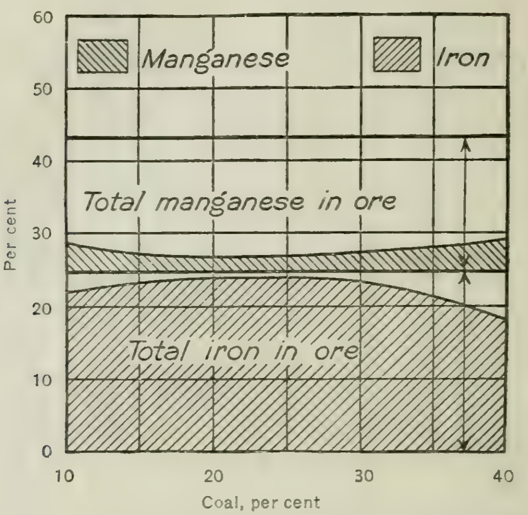

FIGURE 6.—Curves showing percentages of iron and of manganese in magnetic part in heat 34 at 1,250° C. and with varying reducing agent.

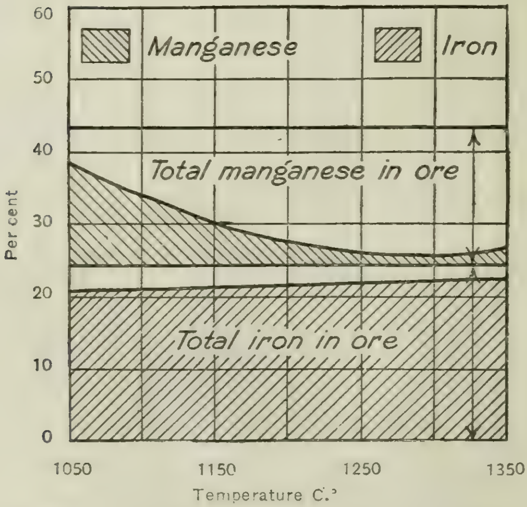

FIGURE 7.—Curves showing percentages of iron and of manganese in magnetic part in heats 16, 17, 31, 32, and 35 with varying temperatures.

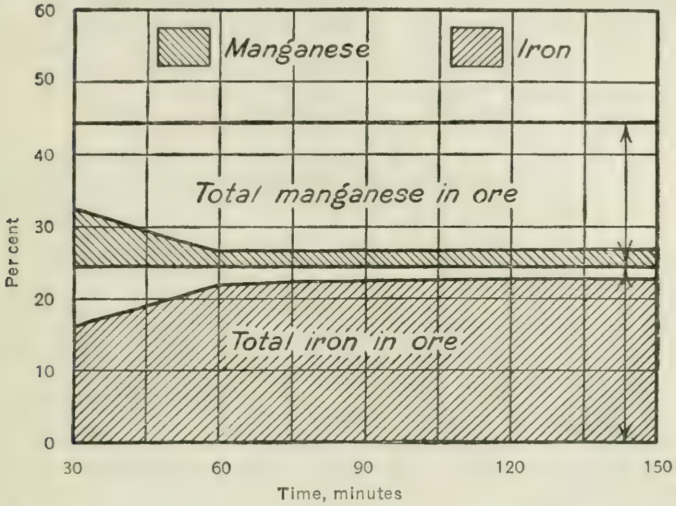

FIGURE 8.—Curves showing percentages of iron and of manganese in magnetic part in heat 35 at 1,250° C.

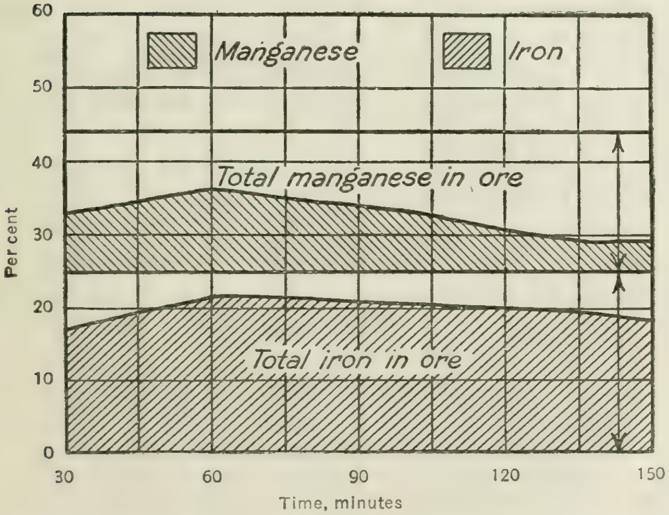

FIGURE 9.—Curves showing percentages of iron and of manganese in magnetic part in heat 17 at 1,150 °C.

CONCLUSIONS.

Figures 2 to 9 indicate that the following are the requisite conditions for the low-temperature reduction of the iron:

(*a*) Time, preferably 30 to 40 minutes, although satisfactory results were obtained up to 2 hours.

(*b*) Temperature, 1,250° C.

(*c*) Reducing agent, 20 per cent coal.

Analyses of ores used in final tests with low-temperature reduction.

Ore.	Fe.	Mn.	SiO_2.	P.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*
West End	24.36	18.12	26.50	0.097
Ferro	28.55	24.55	17.60	.091
1019 E	35.15	13.68	9.46	.179
1016 Cts	40.30	13.08	10.06	.107
381 Cts	35.40	10.25	21.36	.059

Two kilograms of each ore, crushed to 10 mesh and mixed with 20 per cent reducing agent, was charged into a large graphite crucible heated to redness.

The temperature was raised to 1,250° C. and kept as closely as possible between 1,250° and 1,275° C. until reaction was complete.

The time was varied to some extent according to the appearance of the charge, with a maximum time of two and one-half hours, except in heat 43.

The reducing agent used was Elkhorn coal like that used in the preliminary tests, except that in heat 41 coke was used.

The furnace used was a Case oil-fired melting furnace No. 40.

The method of handling the furnace products was practically the same as in the preliminary tests. However, the entire contents of the crucible were poured in a liquid condition, and most of the metal produced was recovered in the form of a large button.

TABULATED RESULTS.

Detailed results of the tests are presented in Table 12 following:

TABLE 12.—*Results of final tests with low-temperature reduction.*

Test No	36	37	38	39	40	41	42	43	44	45	46
Charge.											
Weight:											
Ore ... grams	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000	2,000
Coal ... do	500	400	400	500	400	300	400	400	400	400	400
Lime ... do	...	...	...	...	...	...	...	...	...	...	100
Ore analysis:											
Fe ... per cent	24.36	28.55	(*a*)	35.15	(*b*)	(*b*)	40.30	(*c*)	(*b*)	35.40	(*d*)
Mn ... do	18.12	24.55	(*a*)	13.68	(*b*)	(*b*)	13.08	(*c*)	(*b*)	10.25	(*d*)
SiO_2 ... do	26.50	17.60	(*a*)	9.46	(*b*)	(*b*)	10.06	(*c*)	(*b*)	21.36	(*d*)
P ... do	0.097	0.091	(*a*)	0.179	(*b*)	(*b*)	0.107	(*c*)	(*b*)	0.059	(*d*)
Metal.											
Weight ... grams	463	591	571	927	697	683.60	781	818	703	691	695
Percentage, by weight of ore	24.15	29.55	28.55	36.35	34.85	34.18	39.05	40.90	35.15	34.55	34.75
Analysis:											
Fe ... per cent	92.8	88.90	95.20	95.00	95.90	92.80	96.30	94.50	94.50	94.70	95.60
Mn ... do	2.64	3.90	0.70	1.25	0.56	0.45	0.28	0.84	0.53	0.46	0.35
P ... do	0.4	0.25	0.25	...	0.45	0.50	0.24	0.25	0.46	0.184	0.184
S ... do	0.04	0.003	0.035	0.043	0.057	0.071	0.052	0.027	0.054	0.095	0.079
Recovery:											
Fe ... per cent	87.8	91.8	95.2	98.3	95.0	90.3	93.3	96.0	94.5	93.0	93.6
Mn ... do	3.34	4.7	0.8	3.33	1.4	1.1	0.8	2.6	1.4	1.6	1.2
P ... do	95.0	80.2	80.2	...	81.6	82.7	82.0	93.5	85.0	100.0	100.0
Slag.											
Weight ... grams	1,137	1,056	1,107	851	816	930	785	819	905	955	967
Percentage, by weight of ore	56.9	52.8	55.35	42.55	40.8	46.5	39.3	40.9	45.3	47.7	48.4
Analysis:											
Fe ... per cent	4.9	3.9	2.9	2.2	4.4	6.8	6.4	2.6	4.5	4.6	4.2
Mn ... do	27.3	40.0	42.7	29.2	29.55	26.0	30.8	28.7	27.75	20.6	18.75
Recovery:											
Fe ... do	11.4	7.3	5.6	2.6	5.1	9.0	6.2	2.6	5.8	6.2	5.7
Mn ... do	85.5	86.0	95.0	90.7	88.2	88.3	92.4	89.8	93.5	95.7	88.4

[a] Same as for No. 37.

[b] Same as for No. 39.

[c] Same as for No. 42.

[d] Same as for No. 45.

DISCUSSION.

The foregoing tabulation is self-explanatory. The results were obtained practically under the conditions indicated under preliminary tests. However, the appearance of the charge, and in particular the condition of the slag and metal, were also used to determine the end of the reaction.

PRODUCTS.

On averaging all the results 93.5 per cent of the iron and 2.0 per cent of the manganese contained in the ore were reduced and entered the metal, while 4.3 per cent of the iron and 90.3 per cent of the manganese remained in the slag. The composition of the metal obtained in all the tests varied as follows:

Iron, per cent, 89.9 to 96.3; average, 94.2.

Manganese,[a] per cent, 0.28 to 3.9; average, 1.0.

Phosphorus, per cent, 0.1842 to 0.50, or 88.0 per cent of that contained in the ore.

Sulphur, per cent, 0.003 to 0.097, or about 10 per cent of that contained in the charge.

ORES.

All the ores tested were of the intimately disseminated Cuyuna manganiferous type and represent a wide range of iron and manganese content. All were crushed to pass a 10-mesh screen, except that used in tests 45 and 46, which was crushed to 8-mesh. It is probable that if coarser material were used the time necessary to complete the reaction would increase. The uniform results obtained in testing such a variety of ores would tend to show that this low-temperature reduction is applicable to a much greater variety of ores.

TIME.

A considerable range of time at which the charge was kept at a temperature between 1,250° and 1,300° C. was used. From the data thus obtained it is apparent that, under the conditions given, the best results are obtained when the charge is kept at this temperature for 30 to 40 minutes.

REDUCING AGENT.

Elkhorn coal crushed to 10 mesh was used in all tests, except No. 41, in which foundry coke, crushed to the same mesh, was used. The Elkhorn coal analyzed 6.66 per cent ash, 40.98 per cent volatile matter, and 0.645 per cent sulphur. The coke analyzed 13.9 per cent ash, 3.33 per cent volatile matter, and 1 per cent sulphur.

No difference was noticed in the reaction when coke was used, except that the slag fused at a lower temperature. This was probably due to the larger percentage of ash in the coke. The use of coke also tends to increase the sulphur in the metal. The proportion required did not seem to vary with the character of the ore.

---

[a] Discarding the manganese content of metal obtained in tests 36 and 37, the reactions of which were not completed, the average manganese content of the metal would be reduced to 0.60 per cent.

LIME.

The addition of lime in test 46 did not materially change the results. Its use as a flux would probably be necessary to obtain a high recovery of iron from ores containing small amounts of manganese and large percentages of silica. The lime used was of the commercial air-slaked type.

FINAL TESTS.

OUTLINE.

In the final tests with low-temperature reduction, ores of the following analyses were used:

HIGH-TEMPERATURE REDUCTION TESTS.

GENERAL OUTLINE.

The slag obtained by the process of low-temperature reduction was crushed to pass a 30-mesh screen, mixed with 25 per cent coke crushed to pass a 10-mesh screen, and charged into a carbon crucible which was heated in an ordinary electric carbon resistor furnace.

CHARGE.

An analysis of the slag used for the charge was: Iron, 4.9 per cent; manganese, 28.68 per cent; SiO_2, 31.1 per cent; Al_2O_3, not determined. An analysis of the coke used as a reducing agent is given above. Slag, coke, and lime were mixed before putting the charge into the crucible. The only variable used in these tests was lime. This was air-slaked material similar to that used in the low-temperature final tests.

PYROMETER.

The temperature was measured by means of a Scimatco pyrometer. This instrument, though not very satisfactory, was the best available. It was first calibrated against a Hoskins thermocouple, having a temperature capacity up to 1,100° C. With great care in observing by keeping the comparison lamp at a constant amperage fairly concordant results were obtained. The temperature measured was that of the surface of the charge only, but when a charge is in constant agitation, owing to the carbon monoxide produced by the reaction, it is assumed that the surface temperature serves as a fair indication of the temperature of the crucible contents.

CRUCIBLES.

A number of standard crucibles were tried, as follows: Ordinary assay crucible, graphite crucible, and alundum crucible. None of these could withstand the conditions. The crucibles finally used were cut on a lathe from 4-inch carbon electrodes. With care they would hold 200 grams of crushed slag mixed with the requisite coke and lime. These crucibles suffered deterioration only from oxidation at the upper edge and could be used until they became too shallow to hold the charge.

TEMPERATURE.

The temperature at which a satisfactory reaction took place was found to be about 1,450° C. This satisfactory reaction may be designated as uniform and rapid. Above 1,450° C. the reaction was so rapid that it caused violent boiling, and below this temperature the reaction was so slow and inactive that no appreciable metal was reduced. Hence, the condition of the reaction was used as a criterion for temperature regulation.

TABULATED RESULTS.

The detailed results of the high-temperature reduction tests are presented in the tabulation following:

TABLE 13.—*Results of high-temperature reduction tests.*

Test No	54	55	56	57	58
Charge.					
Weight:					
Slag .... grams	200	200	200	200	200
Coke .... do	50	50	50	50	50
Lime .... do	40	30	20	30	[illegible]
Analysis:					
Fe .... per cent	4.9	(a)	(a)	(a)	(a)
Mn .... do	28.68	(a)	(a)	(a)	(a)
SiO_2 .... do	31.10	(a)	(a)	(a)	(a)
Alloy produced.					
Weight .... grams	66.0	66.3	66.7	76.4	81.5
Percentage of slag charged	33.0	33.15	33.35	37.7	40.75
Analysis:					
Fe .... per cent	16.8	15.40	16.40	12.5	12.0
Mn .... do	66.9	65.00	6.42	62.4	58.4
Si .... do	10.7	13.77	13.56	23.6	25.0
Recovery, per cent of content of slag charged:					
Fe	[b] 112.0	[b] 102.6	[b] 106.0	[b] 97.0	[b] 100.0
Mn	77.2	75.4	74.8	83.2	84.0
Si	24.1	31.4	31.0	62.0	70.5
Slag produced.					
Weight .... grams	134.0	108.7	112.3	83.3	82.9
Percentage of slag charged	67.0	54.4	56.15	36.9	41.45
Analysis:					
Fe .... per cent	1.26	1.94	2.15	1.37	1.30
Mn .... do	6.15	4.25	7.00	4.70	5.44
SiO_2 .... do	36.25	34.85	37.80	38.10	30.60

[a] See test 54 for analysis.

[b] Recoveries are calculated on content of slag charged. Discrepancy shown by getting more than 100 per cent recovery is due to fact that coke contains iron as part of the ash.

DISCUSSION.

GENERAL.

Coke was used as a reducing agent because it was thought to be the best commercial source of carbon suitable for this high-temperature reduction. It was mixed well with the rest of the charge and seemed to remain more or less disseminated through the charge after the slag portion of the charge had melted. The amount used in the tests does not indicate the proportions required on a commercial scale where no other source of carbon would be present, because with the use of carbon crucibles there was always an excess of carbon present from this source.

Lime was added to the charge to render the resulting slag sufficiently liquid to pour. The amount added was varied for the purpose of controlling the reduction of silicon. However, the effect of lime in this respect can not be considered very definite, although the tendency is to decrease the silicon in the alloy as the lime is increased in the charge.

The data obtained indicate that the reduction of silica to metallic silicon increases as the temperature is increased and is a function both of the temperature and of the lime added. Possibly, if the lime were incorporated in the slag portion of the charge, its effect would be more pronounced.

ALLOY.

The metal produced is essentially a silico-ferro-manganese, consisting of about 15 per cent iron, 60 to 67 per cent manganese, and 10 to 25 per cent silicon. The percentage of silicon increases with the temperature of the reaction; and as the silicon increases the percentage of manganese in the alloy will correspondingly decrease.

RECOVERY.

The excess of iron recovered, together with some iron remaining in the resulting slag, is explained by sources of iron being present in the charge, which were not taken into account in making the calculations. The recovery is based on the slag portion of the charge only. Under the given conditions, the manganese remaining in the resulting slag was not less than about 5 per cent. It is, therefore, evident that as the slag volume is decreased the recovery of manganese is increased. This is shown in experiments Nos. 57 and 58. The recovery of manganese, as given in the tabulated results, varied from 75 to 84 per cent, and averaged 79 per cent.

SLAG.

As the iron and manganese is reduced from the charge, the fusibility of the slag decreases and its viscosity increases as a result of the increase in unbalanced silica. The presence of lime in the charge is advantageous, increasing the fusibility of the slag, and decreasing its viscosity, so that it may be handled in a liquid condition.

SUMMARY.

The essential results contained in the foregoing report may be summarized in semitabular form as follows:

Summary of results of experiments with the Jones process.

LOW-TEMPERATURE REDUCTION, 1,250° TO 1,300° C.

Charge:

Reducing agent, 20 per cent of a good grade of coal.

Ore:

Fe, per cent	24.0 to 40.0
Mn, per cent	10.0 to 24.0
SiO_2, per cent	10.0 to 26.0
Al_2O_3, per cent	Not determined.

Products:

	Approximate composition, per cent.	Recovery, per cent.
Liquid metal:		
Fe,	94.00	94.00
Mn,	1.00	2.0
Si,	.10 (maximum)	.....
P,	.50 (maximum)	88.0 (average)
Liquid slag:		
Fe,	5.00	6.0
Mn,	29.00	90.0
SiO_2,	31.00	.....
Loss	Manganese	8.0

HIGH-TEMPERATURE REDUCTION, 1,450° C.

Charge:

Liquid slag from low-temperature reduction.

Coke, 25 per cent of slag.

Products:

	Approximate composition, per cent.	Recovery, per cent.
Liquid metal:		
Fe	15	100.0
Mn	55 to 65	80.0
Si	10 to 25	.....
Liquid slag:		
Fe	1.0 to 2.0	
Mn	4.0 to 6.0	
SiO_2	30.0 to 40.0	

CONCLUSIONS.

The foregoing results seem to indicate that by the Jones process concentration of manganese in the finely disseminated manganiferous iron ores is metallurgically possible. A discussion of the commercial possibilities of the process will not be attempted.

The metal produced in the low-temperature reduction is suitable for conversion into steel by any basic steel process. It could not be used as a foundry metal. Whether the low-temperature reduction could be modified to yield a foundry metal would require further experiments. By a suitable regulation of temperature and by the use of iron ore in the place of manganiferous ores, it is possible that a foundry metal could be produced.

The high-temperature reduction produces an alloy that may be termed a silico-ferromanganese. Although an alloy of this nature would be new to the steel industry, it could probably be used to advantage. About 72 per cent of the manganese in the ore appears in the alloy.

CHAPTER 7.—COST OF PRODUCING FERRO-GRADE MANGANESE ORES.

By C. M. Weld and W. R. Crane.

INTRODUCTORY STATEMENT.

In collecting and disseminating information on methods of mining and preparing domestic manganese ores, careful consideration was given by the bureau to the important matter of costs. Unfortunately few operators kept systematic cost sheets, hence the data collected are incomplete and unsatisfactory, particularly for an industry having so little uniformity. Domestic manganese deposits, especially deposits of ferro-grade ores (35+ per cent metallic manganese), vary widely, not only in the character of the ore, but also in the conditions of occurrence. Costs must therefore be expected to vary correspondingly, even when cost-keeping methods are uniform. But when the methods are as variable as the costs, computation becomes still more difficult.

It has, nevertheless, been possible by carefully compiling the material at hand to arrive at certain estimated average figures which are believed to be approximately correct, and to draw therefrom certain general conclusions. It must be clearly understood, however, that these average figures may be entirely misleading if applied to some one particular deposit. The conditions are so variable that it is necessary to consider each deposit strictly on its individual merits when considering its commercial possibilities. The average figures, however, have distinct value as guides to the probable competition which will have to be faced.

In fact these average figures have been compiled with this latter point especially in view. For this reason the more immediate foreign competitors, Brazil and Cuba, are considered as well as the domestic sources.

The subject may be approached in two ways—by determining so far as possible the actual cost, and by determining what may be called the "proper" cost. It is thoroughly appreciated that at many properties, owing to a multitude of reasons, costs have been unduly and unnecessarily high. However, as regards the costs of producing manganese ores during the war, it would serve no useful purpose to estimate what the cost ought to have been. War costs have become a matter of history, but the actual figures as bearing on probable future costs are of vital importance. When it comes to estimating the future, the opposite angle of approach may be taken. Domestic ores will soon once more have to face active and unrestricted foreign competition, and "proper" costs will have to be attained in order to keep the industry alive. Furthermore, if the question of protection be raised, only "proper" costs should be considered. It is appropriate, therefore, to discuss future possibilities from the second point of view, namely, as regards what the costs should be.

Another vital question, in every discussion of this sort, is what constitutes cost. Strange as it may seem, the important factors of amortization of plant and depletion of ore reserves are more often overlooked than not. This has no doubt been especially true of a business like that of the manganese industry during the war, chiefly characterized by the hurried and ofttimes ill-considered undertakings of many small operators with little financial strength. Nevertheless, owing to the uncertain life of the business, the writing off of the investment should have been the prime consideration.

In most instances the cost data collected evidently do not include allowances for amortization and depletion. It is now obviously impossible to say what these items should have amounted to; therefore, where they do seem to have been included, they will be omitted from the following discussion, for uniformity. This fact, however, should be kept in mind, if any comparison of the average estimated war costs with the prices then prevailing is attempted. The apparent profit which such a comparison shows probably failed, in a large number of cases, to offset the investment before the business collapsed upon the signing of the armistice. In fact, were not this the case, there would be little ground to-day for any claims under the War Minerals Relief act.

The same considerations must be kept in mind as regards the estimated average future costs. These are not intended to include appropriate charges for writing off investment.

COSTS OF DOMESTIC MANGANESE ORES.

The domestic manganese ores of ferro grade fall into two distinct classes, which vary widely from one another in their cost of production. These are (1) the carbonate ores, and (2) the oxide ores. A third class, namely, the silicate (rhodonite) ores, should perhaps be mentioned, but such ores in the natural state are too siliceous, and concentration too costly, to make their use feasible even with a war-time market. Consequently the small amounts produced were incidental to the production of carbonate and oxide ores, and they need not be considered separately.

CARBONATE ORES.

Carbonate ores were mined in quantity in only one locality, namely, Butte, Mont., but the output for 1918 was more than 20 per cent of the total output of manganese ores mined in the United States. The great bulk of the ore required no concentrating and little if any hand sorting. Mining was of the underground type, but the deposits were persistent and the mining operations were efficient. Furthermore the haul to the railroad was short, and accomplished by motor trucks over well-paved roads. It is understood that the cost on board railroad cars was not more than $5 per long ton—probably less. As the product contained 36 to 37 per cent metallic manganese, the unit cost was about 14 cents.

A small amount of siliceous carbonate ore was produced in the Butte district and concentrated at a neighboring customs mill, where the milling cost is said to have been $4.50 per ton of product.

OXIDE ORES.

The oxide ores occur mixed with clay, chert, and other gangue materials and invariably require careful hand cleaning or mechanical washing or concentrating. The deposits are generally erratic. Production costs have therefore necessarily been much higher than with the carbonate ores. In a few places carbonate ores were encountered as the weathered oxide ores were followed downward, but these carbonate deposits were too small and generally too impure, to lessen the otherwise high costs. Occasionally silicate (rhodonite) ores were encountered with depth, when mining as a rule had to be abandoned.

The oxide ores are widely scattered over the United States. They represented nearly 80 per cent of the 1918 production. In estimating the average cost of producing them, they will be treated as a group, although individual costs as between States, districts, and even mines in the same district, frequently varied within wide limits.

These variations were principally due (*a*) to inherent differences in the deposits as regards their size and richness, and (*b*) to their situation as affecting the cost of hauling to the railroad. Costs may best be discussed therefore under two main heads, namely (*a*) cost at the mine, and (*b*) cost of transportation to the railroad.

COST AT THE MINE.

Ordinarily it would be found convenient to subdivide the cost at the mine into mining cost and treatment or preparation cost. A large part of the manganese ore produced, however, was hand cleaned by picking or screening, and the available cost data do not differentiate between mining and hand cleaning. Consequently, three kinds of costs must be considered under mining and preparation, namely, mining, milling, and mining plus hand cleaning.

Figures for these costs as given below include labor, supplies, and the usual overhead charges, such as royalty. It will be of interest to examine these items separately before taking them up as lumped together under mining and treatment.

LABOR.

The labor supply, never abundant in many of the manganese districts, was seriously affected by conditions resulting from the war, not the least of which was the draft. Table 14 following shows data regarding wages recently paid.

TABLE 14.—*Wages paid in several of the manganese-producing States.*

(Dollars per day.)

	Arizonia.	Arkansas.	Georgia.	Montana.	Nevada.	Tennesee.	Utah.	Virginia.	Washington.
Carpenter	....	....	3.25	6.00	....	3.00	....	....	4.00
Common labor	4.75	3.00	2.75	5.50	5.00	2.50	4.00	2.00	....
Underground mines	....	....	3.75	5.50	....	....	....	....	5.00
Open-cut mines	....	3.00	2.75	....	....	....	....	....	4.50
Steam-shovel operator	....	5.75	5.00	....	....	....	....	....	....
Steam-shovel helper	....	2.75	4.00	....	....	....	....	....	....
Dinky driver	....	....	3.75	....	....	....	....	....	....
Fireman	....	2.75	2.75	....	....	....	....	....	....
Mill men	....	3.75	3.00	5.50	....	3.00	....	....	4.50
Hand picking	....	....	2.50	....	....	....	....	....	....
Mine foreman	....	6.00	....	7.50	....	3.50	....	....	....
Mill foreman	....	4.00	....	8.30	....	....	....	....	....
Blacksmith	....	....	3.25	6.00	....	5.00	....	....	5.00

Sufficient data are not at hand for satisfactory comparison between prewar and war times, but a few figures in this direction are available. For example, in 1916 the wage paid common labor in the Cartersville district, Georgia, was $1.25 per 10-hour day; a year later it had risen to $1.75, while in 1918 it was $2.50 per 8-hour day. In certain localities a bonus was paid for a maximum number of days' work per week, which raised the wage to $3 per 10-hour day. A similar condition existed in the Batesville district, Arkansas, and undoubtedly such conditions were universal through the manganese fields of the country. A wage of $2.50 for eight hours is probably a fair average.

SUPPLIES.

No detailed information as to cost of supplies per ton of product is available. The item was far higher, possibly double, the prewar cost.

ROYALTY.

A royalty commonly demanded during the war was 10 per cent of the gross value of the product, that is to say, of the selling price. The royalty thereby varied automatically with the grade of the ore.

Occasionally an arbitrary sliding scale was used, of which the following is an example:

Example of arbitrary sliding scale of royalty for manganese ore.

Manganese content of ore, per cent.	Royalty per ton.
20 or less	$0.50
20 to 30	1.00
30 to 40	1.50
40 and over	2.00

This scale provided for the higher grades of manganiferous ores as well as for ferro-grade ores. At some places, where the ore ran fairly uniformly as to grade, a single fixed rate of royalty was paid, as for instance 50 cents per ton at the Cason mine, Batesville, Ark.; or $1.50 per ton in the Erickson district, Utah.

The average manganese content of all ferro-grade oxide ores mined in the United States was approximately 41 per cent, and the average royalty paid for such ores may be taken to be $1.50 per long ton. Before the war, the current royalty for rather better grade ore was $1.00.

MINING AND TREATMENT.

The mining cost per ton of product naturally varied widely, not only with the cost of handling the bank dirt, or crude ore, but also with the yield of concentrates. As a rule, however, cost and yield roughly offset each other. Thus, a lean bank dirt had to be mined more cheaply than a rich one. Yields varied from 2 or less up to as much as 10 tons of bank dirt per ton of concentrates; and mining costs, where the ore was mined as it came and was sent to a mill for treatment, were found to range from $5 to $10 per ton of product, with the average not far from $7.50.

Milling costs likewise varied with the grade of the bank dirt, as also with the degree of refinement to which the process was carried. Concentrating plants varied from simple log washers, with or without picking belts, to elaborate mills with screens, jigs, and even occasionally tables. Water was not infrequently a costly item. Costs per ton of product varied from $5 to $8.50, the average being approximately $6.50.

At many small operations the bank dirt was simply picked or screened by hand. A profit in such cases was only possible when the crude material was fairly rich, but frequently cheaper costs were attained than at the larger and more elaborate operations, chiefly because overhead charges were reduced to a minimum, and the "mine" was readily moved from point to point, following the richer and cheaper ore. The average cost per ton of product was $12 to $13; individual costs were frequently less.

Considering the respective proportions of ore hand cleaned and milled, the average cost of the total product is estimated to have been $13.50. The figures that contribute to this average include a certain amount of the overhead charge in many cases, but probably $1.50 should be added to cover royalty, sampling, and similar overhead charges, which in other cases have not already been included. Thus it is concluded that the total average cost at the mine of the oxide ores was approximately $15.

The following individual examples are given as illustrating some of the details underlying the above conclusion.

Mr. F. G. Moses of the bureau's staff reported mining costs per ton of product in the Erickson district of Utah as given below. Costs at the mine were comparatively low at that point.

Costs per ton of product in the Erickson district, Utah.

Mining	$5.00
Overhead	2.00
Sampling	1.00
Royalty	1.50
Total	9.50

The average cost of mining per ton of product in the Cartersville district, Georgia, was about as follows:

Average costs per ton of product in the Cartersville district, Georgia.

	Underground.	Open cut.
Mining	$3.00	$2.50
Timbering	1.50	......
Royalty	1.50	1.50
Handling and miscellaneous	2.25	.50
Total	$8.25	$4.50

The bulk of the ore was obtained from open-cut workings and the average cost per ton of product is assumed to be $5.50. This is by far the most important manganese producing district in the State of Georgia.

The following rates paid for contract mining in the Batesvillo (Arkansas) and Cartersville (Georgia) districts are interesting.

Rates for contract mining per ton in Batesville and Cartersville districts.

Ore.	Batesville.[a]	Cartersville.[b]
Low grade	$8	$17
Medium grade	$12 to $15	....
High grade	$25 to $40	....

A high price was paid in Batesville for high-grade ores, yet the amount of ore that could be mined was much less than with low-grade ores, so that the wage earned was about the same.

---

[a] The wash dirt mined to be cleaned at company's expense.

[b] For all lump and cleaned ore.

The following figures are given as approximate milling costs per ton of concentrates at a large operation in Virginia:

1917: 729 tons were cleaned at a cost of $10.77 per ton.
1918: 945 tons were cleaned at a cost of $6.53 per ton.
1918: Estimated cost for remodeled mill, $6.50 per ton.

The total estimated cost per ton of product on board railroad cars, including all charges, was given for the latter part of 1918 as $15. This is only one of a number of relatively important operations in Virginia.

Theodore Simons reported the following average costs for July, 1918, in the Philipsburg district, Montana. These costs are per ton of product on board railroad cars and include all charges for labor, timbering, hand cleaning or concentrating, road construction, development, repairs, and overhead:

Average costs in Philipsburg district.

Mining Company A	$15.14
Mining Company B	21.00
Mining Company C	16.00
Mining Company D	17.33
Mining Company E	17.00
Mining Company F	18.00

The arithmetical average of the above is $17.41 per ton of the product. As company A was by far the largest single producer its lower cost would bear down this figure in a weighted average to approximately $16.50, or roughly $15, excluding the cost of hauling and loading into railroad cars.

A representative milling cost at Philipsburg was reported as being $7.14 per ton of product. Custom mill work on Butte and Philipsburg oxide ores is said to have cost $2.50 per ton of crude ore. As the ratio of crude to concentrates was approximately 2.6 to 1, this latter figure would become about $6.50 per ton of concentrates.

COST OF TRANSPORTATION TO RAILROAD.

The location of the mines worked for manganese during the war with reference to the nearest railroad shipping point varied as widely as the other factors entering into cost. Not infrequently the distance hauled differed greatly even within the same district. For instance, in the Batesville district, Arkansas, hauls were commonly 2 to 3 miles, but hauls of 5 to 10 miles were not uncommon. The ore from a mine in Tennessee was hauled 14 miles to the railroad, and in the Buena Vista Valley, Nev., the distance hauled was 20 miles.

Transportation was by wagons and motor trucks. The average load was 1 ton. This average figure frequently applied even when 2-ton trucks were used, on account of poor roads. A few 5-ton trucks were used, but the tonnage handled in such units was small.

Table 15 following gives cost data and other data relating to a number of typical and important camps. The average cost of hauling for all oxide ores was roughly $3.50 per ton. To this should be added an average charge of 35 cents per ton for handling the ore at the two terminals of the haul.

TABLE 15.—*Cost of handling and hauling ore in manganese-producing States.*

State.	Average distance hauled, miles.	Load hauled, pounds.	Method employed.	Cost per ton.			
				Loading cars.	Hauling to railroad.	Loading wagons.	Weighing ore.
Arkansas	3⅓	3,300	Wagons and motor trucks	$0.25	$2.60		$0.10
Georgia	3.5	2,800	Wagons		1.10		
Montana	[a] 1½	3,000	Wagons and motor trucks	[b] .29	1.45		
Nevada	10	4,000	do		5.95		
New Mexico	18	2,800	Wagons	.45	3.50	$0.35	
Tennessee	2½	3,000	do		2.50		
Utah	13	2,800	do	.50	6.50	.50	
Virginia	2½	4,000	do	1.00	1.50		

[a] Philipsburg district only. [b] Cost of loading both wagons and cars.

SUMMARY OF COST OF OXIDE ORES ON BOARD CARS.

Adding the cost per ton of product at the mine, $15, to the cost for transportation to railroad plus handling, $3.85, the total estimated average cost on board railroad cars for the oxide ores is seen to be $18.85, or, say, $19 per long ton. With 41 per cent average metallic manganese content the unit cost is 46.4 cents.

COST OF RAILROAD TRANSPORTATION TO MARKET.

Railroad freight rates advanced materially during the war. Certain reductions were later made in favor of western shippers of manganese ores, but were still high. Some typical rates are given below.

Freight rates between Southern points and Chicago.

From—	Rate per ton of 2,240 pounds.
Northeastern and northwestern Georgia	$6.90
Eastern Tennessee and western North Carolina	5.90
Virginia	6.30
Abbeville and Greenville, S. C., and Lincolnton, Ga	8.40
Batesville, Ark	4.60

Freight rates between western points and Chicago, Philadelphia, and Birmingham.

State (any point).	Destination.	Rate per short ton.	Destination.	Rate per short ton.
Colorado	Chicago, Ill	$7.00	Philadelphia, Pa	$8.50
Oregon	do	11.00	Birmingham, Ala	12.50
Washington	do	11.00	do	12.50
California	do	11.00	do	12.50
Montana	do	8.00	do	10.50
Arizona	do	9.00	do	9.00
Nevada	do	10.00	do	11.00
Utah	do	9.00	do	10.00
New Mexico	do	7.00	do	7.00

The Government schedule of prices on which ores were sold during the war fixed the price per unit delivered at Chicago. The producer calculated the value of his ore at Chicago and then deducted the freight rate from point of production to Chicago to get the selling price of his ore f. o. b. railroad cars at shipping point. The buyer paid the actual freight from shipping point to point of consumption.

Whether ores are sold in the future f. o. b. cars at the shipping point or delivered at the furnace makes no difference to the producer. It is at the furnace that he must compete with other producers, whether domestic or foreign, and he must therefore compute the freight as part of his cost of production.

A small amount of ferro was made in western electric furnaces during the war, but the great bulk of it was made east of the Mississippi, at Chicago and in Alabama, Virginia, Pennsylvania, and New York. The production of Pennsylvania furnaces largely predominated, and this will unquestionably be even more true in the future, as production outside of Pennsylvania was principally stimulated by war prices.

It has already been pointed out that costs at the mine and costs of transportation to the railroad varied widely from point to point. These variations were not due to the section of the country in which the mines were situated, however, but rather to the nature of the deposits themselves. Two distinct groups of deposits have been differentiated, namely, the carbonates and the oxides. Within these groups it mattered little whether the mine was situated, for example, in Virginia or in California; the cost might be high or low in either place. Therefore it was found desirable to average all costs for each group and so determine an estimated average cost for producing ore of that group, irrespective of its locality.

It seems necessary to consider the matter of railroad freights from another angle. If all oxide ores, for instance, had been produced in Virginia, their average cost delivered at Pennsylvania furnaces would have been less than if a part had been shipped from points west of the Mississippi. Therefore, the freight for each State or district to the principal center should be weighted in an average according to the amount of ore shipped from that State or district. It has been necessary to make certain assumptions, as exact freight rates are in many cases not at hand, but the resulting figures without doubt approximate the truth.

Proceeding by this method and assuming that the principal consuming center is Pennsylvania, it is found that the average freight rate for the oxide ores during the latter part of 1918 was approximately $10.75 per long ton and for carbonate ores approximately $11.25, including war tax.

SUMMARY OF COST OF DOMESTIC ORES.

The foregoing estimates are summarized as follows:

Summarized data on estimated average cost of domestic ores.

	Carbonate ore.	Oxide ore.	Average.
Estimated tons produced in 1918	63,000	231,500	294,500
Estimated cost on railroad cars	$5.00	$19.00	$16.00
Estimated railway freight	$11.35	$10.70	$10.85
Total estimated cost delivered	$16.35	$29.70	$26.80
Manganese, per cent	36.5	41.0	40.0
Estimated cost per unit, cents	45.0	70.7	65.0

Of course, the more favored districts could produce oxide ores more cheaply than the average figures given. Virginia, for instance, probably delivered ore to the furnaces at a cost of about 55 cents per unit. Cartersville (Ga.) and Batesville (Ark.) ores should have reached Birmingham, Chicago, or Pennsylvania furnaces for 55 to 60 cents per unit. Nearly 85 per cent of the oxide ores, however, came from the far western States and were subject to high railway freights. The carbonate ores paid a high railway freight but other costs were so low that these were probably the cheapest ores produced either on a ton or unit basis.

DISCUSSION OF FUTURE POSSIBILITIES.

The average costs computed in the foregoing pages are for war times and more particularly for the last six months before the signing of the armistice.

In estimating future costs we have little to guide us in the way of prewar costs. Then only small amounts of domestic ores were mined, frequently in a small way by farmers when not busy with farm work. The production was in the southeastern States, where freight rates to consuming centers were not excessive. Probably only small profits were made, but these were satisfactory under the circumstances. Costs were $4 to $8 per ton on board railroad cars, but these costs are not fair guides to future possibilities.

Aside from railway freights, the wage scale enters more largely into the cost of production than any other single item; and it is probable that reduction in cost will come chiefly from reduction in wages. Better practice should account for some reduction, and cost of supplies may perhaps be properly expected to drop about in proportion to reduction in wages.

The consideration of future wages is a far-reaching problem, involving infinitely more important national industries than manganese mining, and an infinite number of correlated problems. The subject is ventured upon only because it seems desirable at this time to attempt some reasonable forecast of the probable future cost of manganese.

There is admittedly much difference of opinion in this respect, but it may fairly be assumed that wages must inevitably drop. The cost of labor is now approximately double the prewar figure in many of the more important manganese-producing districts. An instance has already been given where it has more than doubled. Its advance has frequently outsped the increase in cost of living, which is said in general not to have increased more than approximately 50 per cent in most manganese mining camps. The economic pressure of competition must either depress these wages or else the industry must be abandoned. If it be assumed that wages will drop, to fall in line with the increased cost of living, and that the latter will not decrease, then the drop in wages will be equivalent to 25 per cent of the war wage.

In this connection it may be further assumed that the cost of supplies will decrease in like proportion; whereas better practices will prevail and more efficient labor will be available than under war conditions. The net result is an estimated future reduction in cost of manganese ore on board railroad cars of 25 per cent.

It does not seem likely that railway freights will be materially lowered in the near future, except perhaps to the extent of the removal of the war tax.

On the basis of the foregoing premises, the summarized costs for war times given on page 101 become as follows:

Revised estimated average cost data on domestic manganese ores.

	Carbonate ore.	Oxide ore.	Average.
Estimated proportionate tonnage, per cent	21	79	100
Estimated cost on railroad cars	$3.75	14.25	11.72
Estimated railway freight	$11.00	10.40	10.53
Total estimated cost, delivered	$14.75	24.65	22.25
Manganese, per cent	36.5	41.0	40.0
Estimated cost per unit, cents	40.4	60.0	55.6

Here again, in certain more favored districts, as regards freight rates, the oxide ores can undoubtedly be delivered to the furnaces for less than 60 cents per unit. For the Southern States, generally, rates of 50 cents per unit or less should be attained, whereas for Western ores the cost would probably exceed 60 cents. The Butte carbonate ores are the one exception in the far west, owing to exceptionally cheap mining and hauling costs.

COSTS OF FOREIGN ORES.

The principal source of foreign ore during the war was Brazil. The next most important single source was Cuba. Smaller amounts came from other points in the West Indies, Central and South America, and the Orient, chiefly India.

In prewar times, Russia and India were more important sources than Brazil. It seems probable, however, that Russia will not for a few years become a large factor once more, and that the Indian output may also be slow in reaching this country; though English ferromanganese made from Indian ores is likely to appear before long in quantities sufficient to affect the market.

Our more immediate concern is with Brazilian and Cuban ores. Some information regarding costs in these countries is reviewed in the following pages.

CUBA.

Manganese mining in Cuba is generally by open-cut methods. In only one instance has there been anything approaching true underground work. Preparation for market is generally by hand cleaning, though log washers are used at the most important mine in the island. One concentrator, involving screens and jigs, was built and operated during the war.

The price of labor rose rapidly during 1917–18, being at the last as much as $1.80 to $2 per 9-hour day.

Mining and treatment costs naturally varied from point to point, ranging between $5 and $8 per ton of product.

Much of the ore was hauled from mine to railroad by mules or oxen, the cost varying, with the distance, from $2 to $10 per ton. A small amount was even packed on mules at a cost of $10 and upward.

Freight rates to shipping port ranged from less than $1 a ton up to $2.50 and more, according to the length of haul. Charges for loading into ship were sometimes as high as $1.50. Lighters were used in Santiago harbor.

Royalties were advanced materially during the war period. Before the war, $1 per ton was usual, and some leases were made for 80 cents. The figures for 1918 ranged from $1.50 to $2.50. In one instance a sliding scale was used, based on the delivered value of the ore. This scale started at $1,75 for ore worth $35 per ton, and advanced 25 cents for every $5 additional value for the ore. This meant approximately $2 for 40 per cent ore.

The foregoing figures are assembled for convenient comparison in the following table:

Manganese ore costs in Cuba.

Mining and treatment	$5.00 to $8.00
Hauling to railroad	2.00 to 12.00
Cuban railroad freight	1.00 to 2,50
Port costs and charges	0.75 to 1.50
Royalty	1.50 to 2.50

The two largest producers in Cuba were situated, as regards the bulk of their output, directly on the railroad or on tram lines leading to it. This factor must be considered with others in estimating the average cost. The total cost of ore in the ship is taken as $14 per ton, including the above items and some further miscellaneous and general charges. It is, of course, understood that no amortization of investment is included in this figure.

The actual costs at a small operation whose daily output was 12 to 20 tons of hand-cleaned shipping products were as follows:

Detailed costs for one Cuban operation.

Mining and cleaning	$6.50
Hauling to railroad	2.05
Surface rights	.80
Miscellaneous and general	2.00
Railroad freight to Cuban port	2.02
Handling into ship	1.50
Total per ton of product	14.87

The operators owned the ore deposit and therefore paid no royalty. They were heavily penalized, however, by the owners of the surface.

The 1916 costs at another operation, situated on the railroad and at which the ore was washed in log washers, were as follows:

Detailed cost at another Cuban operation, 1916.

Mining and washing	$5.50
Miscellaneous and general	1.00
Surface rights and royalty	1.10
Railroad freight and switching	1.79
Handling into ship	.72
Total	10.11

The costs of this latter operation were probably $3 to $5 higher during 1918.

BRAZIL.

Below is a statement of costs of mining manganese ore at four operations in Brazil.

Cost data from four operations in Brazil.

Item.	Operation.			
	A.	B.	C.	D.
Mining, f. o. b. cars	$5.00	$4.00	$5.00	$5.00
Export tax	1.85	1.85	1.90	1.85
Railway freight rate to Rio de Janeiro	1.50	1.50	[a] 5.35	[a] 5.40
Dockage at Rio de Janeiro	.65	.65		
Lighterage at Rio de Janeiro			2.50	2.50
Transfer from narrow to broad gage railroad			.25	.25
Haulage to station				2.50
	9.00	8.00	15.00	17.50

[a] Companies C and D are handicapped by increased freight rates. The other two companies are still operating under old contracts.

In addition to the above figures a States tax of $3.20 was imposed during the war. Further, royalty charges up to $1.75 per ton were not uncommon. The total range of costs for Brazilian ore into ship would therefore be $12 to $22.50 per ton. As the low cost is for one of the largest and most important mines, it is probably that a weighted average would be approximately $15.

OCEAN FREIGHT RATES.

Freight rates on manganese ores from Brazil, Cuba, and India to the United States and England are given below:

Freight rates on manganese ores by vessel to United States and England.

Pre-war,	India to the United States	$4.32
Do.	India to England	3.84
Do.	Brazil to the United States or England	2.88
Do.	Cuba to the United States	1.50 to 2.50
1916,	Brazil to the United States	5.50 to 6.50
1918,	Brazil to the United States	15.00
1918,	Cuba to the United States	9.50

The rates of $9.50 and $15 per long ton of manganese ore may be taken as representative, respectively, for Brazil and Cuba during 1918. These figures are just about five times the pre-war rates. It is understood that they include insurance.

The average rail freight during the war from Atlantic seaboard to consumer was probably about $3 per long ton.

SUMMARY OF COST OF FOREIGN ORES.

Assembling the foregoing figures, we have a total estimated average cost for ore delivered at the furnace in 1918 of $23.50 for Cuba and $30 for Brazil. On a unit basis this would be 62 cents for Cuba (for ore containing 38 per cent manganese) and 67 cents for Brazil (for ore containing 45 per cent manganese).

DISCUSSION OF FUTURE POSSIBILITIES.

Before the war such ore as was shipped from Cuba was of a higher grade than that mined during the war, and was sold at the Atlantic seaboard for $10 to $12 per ton, or for, say, $8 to $10 per ton into ship at a Cuban port. This was very close to the average cost at that time. It is not likely that the cost of ore from Cuba will drop in the future much below $11 to $12, but it is certain that ocean freights will drop materially, not improbably to the pre-war figure.

On this assumption we may expect to see Cuban ores reach the United States Atlantic seaboard in the not remote future at a cost of about $13.50, or, say, $16.50 a ton delivered at the furnace if the United States rail freight is not reduced. The quality of the ore, in order to be saleable, will have to be better than it was during 1918. If it contains an average of, say, 41 units of manganese, the estimated future cost delivered to the furnace becomes 40 cents per unit.

The Brazilian problem is somewhat more complicated, as it is uncertain what will be done by the Brazilian Federal and State

Governments with regard to the present high taxes. Other Brazilian costs will probably not be much reduced. If we assume that the taxes, which in 1918 amounted to practically $5 per ton, are reduced two-thirds, the estimated average cost of $15 becomes $11.67, or say $11.50 to include some possible slight reduction in other directions.

It is interesting to compare this estimate with the selling price of Brazilian ores at Rio de Janeiro in former years, which were as follows:

Selling prices of Brazilian ores at Rio de Janeiro, 1914–1918.

Year.	Selling price per ton, ries.	Approximate equivalent.
1914	22, $000	$7.50
1915	29, $000	10.00
1916	55, $000	13.50
1917	93, $000	23.25
1918	117, $000	29.25

The real difference in the future will come in the matter of ocean freights. If these drop to nearly their former level, it should be possible to deliver Brazilian ores to the United States consumer for approximately $18.50 per ton. This figure includes $13.50 for cost into ship at Rio de Janeiro, $3 ocean freight, $1, say, for insurance, and $3 for United States rail freight to the furnace.

Brazilian ores reaching this country were formerly of higher grade than those imported during the war. If the grade now improves slightly, say to 46 per cent manganese, the estimated average future unit cost becomes 40 cents.

Thus it may fairly be expected that both Brazilian and Cuban ores can before long be delivered to the United States consumer for about 40 cents per unit. At an equal price, Brazilian ore will always have the preference owing to the higher content of metallic manganese.

COMPARISON OF DOMESTIC AND FOREIGN COSTS.

The estimated average costs for manganese ores, per long ton delivered to the consumer, are compared in the following table:

Estimated average costs of manganese ores delivered to the consumer during 1918 and in the future.

Kind of ore.	Estimated average cost during 1918.			Estimated average cost in the future.		
	Cost per long ton.	Average manganese content, per cent.	Cost per unit, cents.	Cost per long ton.	Average manganese content, per cent.	Cost per unit, cents.
Domestic:						
Carbonates	$16.35	36.5	45.0	$14.75	36.5	40.4
Oxides	29.70	41.0	70.7	24.65	41.0	60.0
Foreign:						
Cuba	23.50	38.0	62.0	16.50	41.0	40.0
Brazil	30.00	45.0	67.0	18.50	46.0	40.0

This comparison indicates that the oxide ores could not compete with foreign ores even when protected by high ocean freights. Had foreign ores moved freely and been sold at strictly competitive prices, the domestic manganese industry could not have developed except perhaps in certain more favored districts in the Southern States, and in the Butte district (carbonate ores). The urgent need for diverting shipping into other channels, with the consequent restrictions and embargoes on imports, together with the fixing of high prices in order to stimulate production to replace such imports, created highly artificial conditions which practically removed the element of competition. In short, the ores had to be had "at any price."

The comparison further shows that when ocean freight rates return to more nearly their normal level, even the most favored oxide ores, as well as the carbonate ores, will have hard going in a competitive field.

Looking into the more remote future, it is not impossible that the costs of Cuban and Brazilian ores will have to be reduced still further in order to compete with Indian and Russian ores, thereby placing domestic ores at a still greater disadvantage unless domestic costs are correspondingly decreased.

It must furthermore be remembered that at the same price per unit the higher grade ore will always have the preference. This fact is reflected in the invariable sliding scale of prices, the higher price per unit being paid for the higher grade ore. In smelting manganese ore, not only does it cost money to melt and remove the impurities as slag, but the loss of manganese is more or less directly proportionate to the volume of the slag. Domestic ores, being generally of lower grade than foreign ores, command a lower price.

Other factors which militate against domestic ores in competition with foreign ores are that they are variable in grade while foreign ores are far more uniform, and their production is irregular while the foreign supplies are generally dependable.

CHAPTER 8.—PRODUCTION OF MANGANESE ALLOYS IN THE BLAST FURNACE.

By P. H. ROYSTER.[1]

INTRODUCTORY STATEMENT.

This chapter gives data collected by the Bureau of Mines on the operation of blast-furnace plants in the United States producing manganese alloys. This information should be of value to the furnace operator; certainly, without necessarily agreeing with any generalizations and conclusions introduced, he can determine from the data assembled in Table 17 what has been done by others with raw materials and furnace conditions approximating his own, provided comparable combinations can be found in the results presented.

The operation of a blast furnace even for making pig iron is highly empirical. Past furnace records and previous experience are nearly as necessary to successful operation as good coke and a hot blast. As in the past, the production of manganese alloys in this country has been comparatively small, and most of the men in charge of manganese furnaces have had relatively little experience on such ores. Hence, the information obtainable from the few records extant is presented herein in considerable detail, to compensate in part for the paucity of data that can be consulted by furnace operators.

The attempt has been made to have this investigation cover the widest possible range of materials and operating conditions. Data from 18 furnaces will be found in the tables following. The information was obtained in the summer of 1918, and includes every furnace in blast that was making manganese alloys, with the exception of the furnaces of the Carnegie Steel Co., and of the Lavino Co., which were unwilling to cooperate in the investigation. The companies which by their generous cooperation have made the report possible are the American Manganese Co., Bethlehem Steel Co., B. & B. Trading Co., Buffalo Union Furnace Co., Colorado Fuel and Iron Co., John B. Guernsey & Co., Donner Steel Co., E. E. Marshall, Miami Metals Co., New Jersey Zinc Co., Seaboard Steel & Manganese Corporation, Southeastern Iron Corporation, and Wharton Steel Co.

---

[1] Assistant physicist of the Bureau of Mines, with headquarters at the mining experiment station at Minneapolis, Minn. The investigation reported in this chapter was made in cooperation with the School of Mines of the University of Minnesota.

FURNACES INVESTIGATED.

The 18 furnaces investigated have been assigned letters and are designated throughout this report by these letters. The physical dimensions of 14 of these furnaces are given in Table 16 following.

TABLE 16.—*Dimensions of 14 of the 18 blast furnaces investigated.*

[All dimensions in feet.]

Furnace.	Distance from stock line to tuyères.	Bosh diameter.	Hearth diameter.	Stock-line diameter.	Distance from tuyères to hearth bottom.	Bosh height.
A	62.1	13.4	8.8	(?)	5.9	8.8
B	55.0	16.5	9.5	(?)	(?)	11.2
C	66.2	14.3	9.5	10.0	5.0	9.3
D	55.8	14.8	9.5	10.0	8.3	17.7
E	63.5	16.0	10.0	10.0	6.5	(?)
F	57.8	15.2	10.5	12.5	7.6	12.5
G	63.0	18.0	11.5	13.5	8.3	12.1
H	53.6	17.0	12.5	13.0	5.7	12.8
I	55.3	18.5	12.5	12.5	7.6	12.0
J	67.4	18.3	13.0	12.5	15.9	12.5
K	63.0	18.6	13.2	14.5	6.0	14.0
O	(?)	17.7	11.0	13.0	(?)	11.9
Q	65.5	17.5	12.5	12.9	6.3	12.5
R	66.0	19.0	13.0	13.0	6.0	12.5

COLLECTION OF DATA.

CHARACTER.

On every visit of the Bureau of Mines field party to the different furnaces the companies have opened for inspection all the furnace records, charge sheets, and chemical analyses in their possession. Periods of ten days' continuous operation were selected during which the various factors of blast temperature, silicon in the alloy, manganese in the slag, etc., were relatively constant, and all the figures applicable to such a period were copied and averaged. The average figures for such 10-day periods constituted the data on a single "experiment" or "run." The guiding principle in the selection of the experimental periods was to cover the extreme range of furnace conditions; for example, operation with high and low grade ores, and with both high and low blast temperature.

COMPLETENESS.

The information obtainable without undue trouble from a blast furnace includes the chemical analyses, the physical condition, and the weights of the ores, stone, coke, coal, scrap, and wind charged and of slag, metal, dust, gas and scrap produced, with the temperature observable in the hearth, the temperature of the blast entering the furnace, and the temperature of the metal, slag, and gas coming out of the furnace. Even with the aid of such data, it is seriously to be doubted whether anything like a complete description, of what takes place in a furnace can be given. In the present investigation, the data obtained fell short of what must be considered essential to a complete understanding of the smelting problem. Information regarding gas analyses, moisture in the blast, the amount and the analysis of the dust blown from the furnace was not obtainable in any case. Analyses of coke, by far the most important single analysis in the whole group, were rather fragmentary. Furthermore, the custom of limiting the slag analysis to a determination of silica, alumina, and manganese is widespread. Naturally no measurements of metal, slag, or tuyère temperatures had been made by the companies; nor had any gas analyses been taken, save in one case. The bureau's field party made a limited number of such temperature measurements at six furnaces and determined also the CO and CO_2 contents of the gas at five furnaces. On the whole, however, the information was far from complete.

ACCURACY.

Aside from purely analytical errors, the applicability of any given analysis is questionable for two reasons: (*a*) The sample analyzed may not have been representative of the pile or car from which it was taken; (*b*) the furnace records failed to give any connecting link between car and pile analysis and the material charged on any given day. Coke was charged by volume and not by weight, making difficult an estimation of the weight of carbon charged. It is obvious that greater accuracy would be obtained from experimental periods taken over a whole month or, better, six months. It must be remembered, however, that differences in operating conditions were being sought and long-period averages flatten out most of the variations from which it was hoped to obtain information.

In a number of runs, it was possible to obtain samples of slag, metal, stone, and ore, which were reanalyzed by the bureau's chemists. No detailed comparisons are given here but the combined effect of all the differences in the analyses introduces in the manganese balance an average uncertainty of about 15 per cent. Even if, therefore, the blast furnace were a precise and sensitive heat engine with a number of definite relations existing between its operating quantities, it is not to be hoped that these relations could be determined from the kind and quantity of data collected here.

TABULATED OPERATING DATA.

Data on ferromanganese production were obtained from 11 furnaces, and are presented in Table 17 in the form of 40 10-day periods. Similiar data from 10 furnaces making speigeleisen are presented in Table 18, in the form of 30 10-day periods. The data in these tables are classified under 23 items as applied to each of the 70 periods.

Items 1, 2, 2a, and 3 give figures on the ore, coke, coal, and stone charged, the units being in pounds per ton of 2,240 pounds of alloy made. Items 4, 5, and 6 give a partial analysis of the ore mixture. Item 7 shows the tons of alloy made in 24 hours (not including scrap made). Item 8 gives the slag weight as calculated by the bureau and is a rather uncertain quantity. Two methods of calculating the weight of the slag, both of which should give the same answer, are as follows:

Method 1.

If a=per cent base in stone,
b=weight of stone,
c=per cent base in slag;
assuming there is no appreciable base in the coke or ore, then

$$\text{Slag weight}=\frac{ab}{c} \qquad (1)$$

Method 2.

If d=per cent silica in ore
e=weight of ore
f=per cent silica in stone
g=weight of stone
h=per cent silica in coke
i=weight of coke
j=0.478 × per cent silicon in metal
k=per cent silica in slag, then

$$\text{Slag weight}=\frac{de+fg+hi-j}{k} \qquad (2)$$

On account of the various errors and omissions in the weights and analyses involved, it is unusual to find any close agreement between the results of the two equations; in some computations the slag weight may be given as 2,500 pounds by equation 1 and as 3,500 pounds by equation 2. Each value under item 8 is the average of the two answers resulting from equations 1 and 2.

Metal analyses for Mn and Si are given by items 9 and 10. The largest disagreement in checking manganese in the metal was 10 per cent, and in checking Si, 1.1 per cent. Values for the temperature of the blast and of the top gas appear in items 11 and 12. For a given furnace, these have some significance. A comparison of figures for different furnaces should be made cautiously, however. At some furnaces, the pyrometer will indicate a blast temperature of 1650° F. with a black blowpipe; at others the pyrometer will show only 1350° F. with a red-hot blowpipe.

Still less importance should be attached to the wind blown per minute as given by engine displacement in item 13. In period 1 on furnace A 12 pounds of carbon per minute was charged and 12,870 cubic feet of air per minute indicated. This is 105 cubic feet of air per pound of carbon. Even with no carbon blown out as dust, none taken up by the metal, and none absorbed in the stack, this means a tuyère combustion producing 35 per cent CO_2, which is impossible.

Items 14 to 18 show the slag analysis. In a number of calculations the lime and magnesia had to be computed together by difference. Item 19 gives the fixed carbon in the coke and coal charged per ton of metal. Items 20 to 23 give the manganese balance. Item 23 shows the percentage of the manganese not accounted for by that in the metal and in the slag. This figure includes practically every error in weights, analyses, and computations in the preceding 22 items and is undoubtedly the least satisfactory figure in the table:

TABLE 17.—*General data from operation of ferromanganese blast furnaces.*

Period No.	1	2	3	4	5	6	7	8	9	10	11	12	13
Furnace	A	A	A	A	A	A	A	A	A	B	B	B	B
1 Ore ....pounds per ton..	5,981	5,303	5,870	6,000	5,090	5,475	5,720	6,166	6,090	7,560	6,212	7,545	6,025
2 Coke ....do....	4,658	5,175	6,290	7,133	5,016	5,628	5,595	7,352	7,836	8,070	7,776	8,435	7,534
3 Stone ....do....	1,980	1,780	1,826	2,320	1,632	1,806	1,860	2,172	2,112	2,700	2,180	2,475	2,265
Ore analysis:													
4 Mn ....per cent..	41.18	39.11	39.13	42.01	42.25	39.80	38.45	36.66	37.14	35.75	36.70	36.00	38.00
5 Fe ....do....	4.54	6.65	7.42	3.33	6.07	11.28	14.95	4.60	3.69	6.35	4.41	6.14	3.43
6 SiO₂ ....do....	7.78	5.64	6.33	8.45	6.73	4.34	2.58	6.14	6.65	15.06	17.05	14.97	15.28
7 Ferromanganese.........tons per day	45.7	37.2	29.0	33.3	35.7	40.9	41.5	29.5	31.2	33.8	35.0	33.8	34.7
8 Slag ....pounds per ton.	2,671	2,115	2,805	3,035	2,248	2,180	2,070	3,335	3,460	4,630	4,110	4,460	3,670
9 Mn in alloy ....per cent..	78.2	70.6	71.3	80.90	73.5	69.4	68.3	78.3	80.35	72.34	77.13	72.88	77.7
10 Si in alloy ....do....	.76	1.17	.92	1.24	1.24	.73	.85	.70	1.86	.62	.58	.74	1.43
11 Blast temperature ....°F..	1,308	1,268	1,196	1,232	1,132	1,279	1,303	1,175	1,150	786	770	886	782
12 Top temperature ....°F..	935	1,025	1,049	1,051	998	1,147	1,103	1,014	980	....	....	....	....
13 Air per minute..............cubic feet..	12,870	12,820	....	....	12,510	....	....	....	....	....	....	....	....
Slag analysis:													
14 CaO ....per cent..	27.2	24.1	23.9	30.1	26.5	30.1	33.1	38.2	50.2	39.5	44.6	39.1	44.3
15 MgO ....do....	9.7	8.9	8.6	10.3	9.6	10.9	11.5	11.8					
16 Al₂O₃ ....do....	13.0	16.2	16.2	13.2	13.5	17.0	15.4	11.1	15.7	11.1	12.5	11.0	14.1
17 SiO₂ ....do....	27.8	28.2	26.3	29.8	28.7	27.9	28.3	25.5	26.2	29.9	28.7	30.5	29.8
18 Mn ....do....	14.6	15.6	18.9	11.7	14.9	9.4	9.4	6.9	5.3	13.6	9.5	13.3	9.1
19 Carbon ....pounds..	3,810	4,140	5,360	5,820	4,060	4,560	4,480	5,992	6,380	7,090	6,810	7,400	6,600
20 Mn charged ....do....	2,462	2,075	2,316	2,520	2,149	2,180	2,200	2,260	2,600	2,700	2,280	2,715	2,292
21 Mn to alloy ....per cent..	71.2	76.4	69.0	72.0	76.3	71.2	71.0	77.8	66.8	60.0	75.8	60.1	76.0
22 Mn to slag ....do....	15.8	18.4	22.9	14.0	15.5	9.5	9.4	10.1	7.0	22.5	17.1	21.9	14.6
23 Mn lost in stack ....do....	12.9	5.2	8.1	14.0	8.0	19.3	19.6	12.0	23.7	17.5	7.1	18.0	9.4

Period No.	14	15	16	17	18	19	20	21	22	23	24	25	26
Furnace	B	F	F	I	G	G	I	I	J	J	J	J	K
1 Ore . . . pounds per ton	8,108	7,099	4,813	4,831	7,525	6,050	8,150	5,850	8,560	5,040	4,900	4,960	4,040
2 Coke . . . do	9,510	7,539	6,092	6,431	5,460	4,863	7,282	4,652	6,940	500	4,900	4,955	6,275
2a Coal . . . do					546	459				432	415	419	
3 Stone . . . do	3,320	3,168	2,091	2,084	2,941	2,675	3,160	2,165	2,516	1,216	126	1,255	1,985
Ore analysis:													
4 Mn . . . per cent	34.00	42.91	43.68	45.76	32.53	32.33	39.26	41.64	34.83	41.73	45.20	43.75	50.65
5 Fe . . . do	7.75	4.09	6.73	5.90	5.60	5.50	2.40	5.51	4.00	4.06	3.80	4.18	7.27
6 SiO_2 . . . do	17.84	11.30	6.87	8.44	12.62	12.62	15.54	7.43	14.92	2.60	2.53	2.86	6.52
7 Ferromanganese . . . tons per day	35.0	32.0	44.9	44.7	56.5	60.7	50.0	82.7	60.8	94.1	70.0	97.9	46.9
8 Slag . . . pounds per ton	5,740	4,100	2,470	2,610	4,600	3,765	5,040?	2,555	6,230	2,050	2,020	1,985	2,190
9 Mn in alloy . . . per cent	71.5	76.3	79.1	78.8	78.2	67.5	71.4	77.1	69.1	79.2	79.7	79.3	75.9
10 Si in alloy . . . do	.58	.37	.83	1.12	1.53	1.61	1.92	.56	2.05	0.75	0.63	0.61	2.43
11 Blast temperature . . . °F	830	735	993	1,027	1,140	1,183	952	1,277	1,108	1,156	1,257	1,172	950(?)
12 Top temperature . . . °F		757	328	143	730	780	1,016	934	849	496		562	
13 Air per minute . . . cubic feet		14,380	22,500	22,500	17,500	16,900	17,100	16,500	20,000	15,900	20,000	19,500	
Slag analysis:													
14 CaO . . . per cent		28.4	35.0	33.3	25.2	26.2	26.6	31.4	24.2	26.0	27.3	27.9	46.4
15 MgO . . . do	35.5	6.5	9.9	11.2	10.7	12.8	14.1	12.6	11.1	8.9	7.1	7.1	1.7
16 Al_2O_3 . . . do	10.7	11.1	10.3	11.5	11.5	13.9	15.2	11.3	11.4	24.2	23.5	24.5	14.7
17 SiO_2 . . . do	31.3	29.1	27.5	27.8	29.8	30.9	27.7	26.0	30.4	21.7	21.3	20.9	27.3
18 Mn . . . do	15.6	19.8	13.2	11.3	13.3	11.6	11.4	12.0	15.1	8.1	13.3	12.1	5.4
19 Carbon . . . pounds	8,340	6,220	5,030	5,310	5,140	4,563	5,990	3,825	5,550	4,395	4,340	4,434	5,420
20 Mn charged . . . do	2,760	3,043	2,100	2,218	2,450	1,969	3,200	2,435	2,985	2,256	2,226	2,168	2,048
21 Mn to alloy . . . per cent	60.6	56.1	84.4	80.2	71.5	77.0	52.7	71.0	51.6	78.7	80.3	81.6	83.0
22 Mn to slag . . . do	32.4	25.6	15.5	13.3	21.9	21.8	18.0	12.6	31.2	7.4	12.1	11.1	5.8
23 Mn lost in stack . . . do	7.0	18.3	.1	6.5	3.6	1.2	29.3	16.4	17.2	13.9	7.6	.7.3	11.2

TABLE 17.—*General data from operation of ferromanganese blast furnaces*—Continued.

Period No. Furnace	27 K	28 K	29 K	30 L	31 L	32 L	33 L	34 L	35 L	36 P	37 P	38 C	39 M	40 M	Average.
1. Ore .... pounds per ton	4,480	4,650	5,215	6,770	6,565	5,312	6,222	5,465	5,200	7,000	4,117	5,245	6,586	6,598	5,992
2. Coke .... do	5,802	6,368	6,329	7,458	8,459	6,500	6,421	4,855	5,164	5,350	5,965	5,704	7,118	7,100	6,320
2a. Coal .... do	....	....	....	660	747	524	424	471	352	545	506	811	998	957	200
3. Stone .... do	2,268	2,184	2,386	2,965	3,100	2,582	2,528	2,200	2,155	3,591	1,686	2,773	3,042	3,510	2,349
Ore analysis:															
4. Mn .... per cent	51.13	50.11	46.94	39.97	39.68	35.45	35.38	32.38	33.53	38.70	45.50	40.70	45.11	46.36	40.33
5. Fe .... do	7.10	7.30	10.84	3.51	3.69	9.54	4.50	8.02	7.81	4.25	5.55	7.45	3.66	4.15	5.93
6. SiO_2 .... do	6.35	5.57	6.24	7.53	7.40	8.91	6.28	10.11	8.63	15.89	4.85	7.01	6.59	4.08	8.60
7. Ferromanganese .... tons per day	85.8	49.5	77.5	48.1	43.0	64.7	50.5	77.2	77.1	61.3	78.0	35.0	42.0	36.8	51.7
8. Slag .... pounds per ton	2,580	2,315	3,285	3,126	3,365	2,886	2,727	2,452	2,256	4,468	1,880	2,700	3,900	3,480	3,196
9. Mn in alloy .... per cent	74.8	75.9	62.5	80.7	81.1	69.8	78.4	70.4	71.8	78.1	80.9	74.9	75.3	74.5	74.9
10. Si in alloy .... do	0.29	1.82	0.26	1.29	1.47	0.75	2.72	1.46	2.22	....	....	2.19	1.20	1.30	1.15
11. Blast temperature .... °F	970	950(?)	998	1,065	1,043	1,089	1,204	1,457	1,207	1,450(?)	1,450(?)	1,387	1,242	1,182	1,135
12. Top temperature .... °F	965	....	845	1,162	1,152	1,162	1,127	1,123	1,140	750(?)	750(?)	52	905	910	690
13. Air per minute .... cubic feet	17,500	....	19,100	17,800	17,700	18,800	17,800	18,800	18,800	....	....	....	....	....	....
Slag analysis:															
14. CaO .... per cent	37.1	46.6	33.3	40.4	47.7	42.0	43.3	41.9	44.3	41.0	45.6	29.8	38.8	43.3	41.75
15. MgO .... do	1.1	1.5	1.4	6.8	5.7	5.6	4.3	6.3	4.9			18.2			
16. Al_2O_3 .... do	11.4	13.8	11.1	11.7	12.8	11.8	14.5	10.5	13.2	14.9	18.8	10.7	14.4	11.1	14.0
17. SiO_2 .... do	28.8	27.6	26.5	31.1	27.4	29.9	27.3	31.6	31.0	35.4	20.3	32.2	25.7	28.7	28.1
18. Mn .... do	14.3	5.7	19.6	8.8	4.6	7.2	6.9	5.8	4.4	4.5	9.9	7.3	15.2	11.8	10.6
19. Carbon .... pounds	4,690	5,450	5,280	6,740	7,655	5,951	5,650	4,432	4,568	5,060	4,700	5,280	6,500	6,471	5,323
20. Mn charged .... do	2,288	2,330	2,442	2,710	2,608	1,888	2,200	1,770	1,745	2,718	2,060	2,138	2,970	3,056	2,382
21. Mn to alloy .... per cent	73.1	73.0	57.2	66.9	69.7	82.8	79.7	89.0	92.1	64.5	88.0	78.7	57.1	54.5	72.0
22. Mn to slag .... do	16.2	5.7	26.3	10.3	5.9	10.1	11.1	8.1	8.7	6.4	4.9	8.9	20.0	13.4	14.7
23. Mn lost in stack .... do	10.7	21.3	16.5	22.8	24.4	7.1	9.2	2.9	−0.8	29.1	7.1	12.3	22.9	32.1	12.8

TABLE 18.—*General data from operation of spiegeleisen blast furnaces.*

Period No.	41	42	43	44	45	46	47	48	49	50	51	52	53	54	55
Furnace	C	C	C	C	D	D	D	D	D	E	E	E	E	E	H
1. Ore..........pounds per ton..	5,192	5,795	5,488	5,365	4,510	4,505	4,516	4,420	4,282	5,930	6,560	6,945	6,550	6,800	4,525
2. Coke....................do....	3,327	3,831	2,923	3,150	3,110	3,180	3,060	3,185	3,200	3,600	4,590	4,640	4,518	4,100	2,970
2a. Coal....................do....	749	926	725	693	....	....	....	....	....	....	....	....	....	....	....
3. Stone....................do....	1,568	2,052	1,655	2,328	1,436	1,691	1,721	1,684	1,530	2,981	3,290	3,090	3,310	3,380	1,331
Ore analysis:															
4. Mn..................per cent..	12.72	12.77	12.77	12.77	14.62	13.62	14.13	13.84	14.06	13.35	12.55	11.27	12.29	12.91	14.85
5. Fe....................do....	36.72	36.90	37.65	37.65	40.20	39.90	37.83	39.33	41.46	31.87	30.84	34.71	32.27	32.18	40.68
6. SiO_2....................do....	17.46	16.65	17.41	17.41	9.78	10.20	10.22	10.55	9.09	19.80	20.81	21.13	22.30	20.20	9.62
7. Spiegeleisen.......tons per day..	44.7	30.2	46.0	51.0	88.3	95.0	95.8	94.7	74.9	83.0	73.0	64.0	66.0	69.0	113.3
8. Slag............pounds per ton..	2,614	2,990	2,330	2,950	2,035	2,050	2,155	2,135	1,940	3,765	4,615	4,400	4,755	4,430	1,870
9. Mn in alloy............per cent..	14.9	16.50	16.2	17.1	19.9	19.2	18.9	17.6	19.4	16.7	14.7	15.7	14.6	16.7	20.0
10. Si in alloy..................do....	3.1	4.9	2.1	1.2	2.3	2.6	1.0	1.3	1.3	0.7	0.8	2.4	0.8	1.1	0.9
11. Blast temperature...........°F..	1,234	1,304	1,344	1,211	1,178	1,166	1,132	1,194	1,100	983	914	1,068	914	1,005	1,208
12. Top temperature...........°F..	613	749	540	538	700	600	500	400	393	....	....	593	....	....	....
13. Air per minute........cubic feet..	8,660	8,740	8,660	8,800	....	....	....	....	15,660	....	....	....	....	....	20,600
Slag analysis:															
14. CaO.................per cent..	26.0	24.3	27.8	31.6	26.1	26.2	25.5	25.8	25.4	42.0	39.3	39.5	37.1	42.0	25.1
15. MgO....................do....	10.1	11.1	12.6	11.6	15.2	15.4	15.0	13.2	14.4						14.1
16. Al_2O_3....................do....	10.3	13.3	9.8	7.8	17.6	16.9	15.2	15.1	16.2	9.4	10.9	13.2	10.9	9.9	14.7
17. SiO_2....................do....	28.3	37.2	36.3	37.2	31.1	31.0	30.4	31.3	30.3	36.6	35.8	37.3	38.1	36.9	30.7
18. Mn....................do....	10.79	10.07	10.48	8.91	6.39	6.31	9.95	10.66	9.35	8.40	9.88	6.73	9.88	7.73	9.68
19. Carbon....................pounds.	3,292	3,740	2,870	3,045	2,820	2,880	2,780	2,890	2,910	3,000	3,830	3,870	3,760	34.20	2,710
20. Mn charged................do....	660	740	695	681	660	614	638	612	601	793	823	780	802	879	672
21. Mn to alloy............per cent..	49.0	50.6	53.4	55.1	67.5	70.0	66.1	61.3	72.2	47.2	40.0	46.4	40.7	42.1	66.6
22. Mn to slag..................do....	42.6	40.7	35.0	38.7	19.7	21.1	33.5	37.2	30.2	39.8	55.4	38.0	58.0	39.0	26.9
23. Mn lost in stack............do....	8.4	8.7	11.6	6.2	12.8	8.9	0.4	−1.5	−2.4	13.0	4.6	15.6	1.3	18.9	6.5

Period No.	56	57	58	59	60	61	62	63	64	65	66	67	68	69	70	Average.
Furnace	H	H	J	J	M	M	M	M	P	B	Q	Q	R	R	R	
1. Ore ..... pounds per ton	5,100	4,300	5,710	5,000	5,025	4,625	5,145	5,665	4,131	5,322	6,050	5,852	5,495	5,818	5,355	5,520
2. Coke ..... do	2,980	2,870	4,900	4,210	3,506	3,297	3,110	3,987	3,290	4,220	5,360	5,147	5,853	7,147	5,048	3,950
2a. Coal ..... do	....	....	280	260	....	238	575	830	340	....	....	....	....	....	....	187
3. Stone ..... do	1,423	1,290	2,155	1,980	2,104	2,026	1,882	1,762	1,450	1,830	3,491	3,921	3,728	3,946	3,480	2,324
Ore analysis:																
4. Mn ..... per cent	14.50	15.53	9.02	11.86	18.8	13.3	11.1	15.95	12.76	11.05	11.10	9.83	10.25	9.86	99.06	12.75
5. Fe ..... do	37.67	40.92	31.60	31.84	31.8	32.3	35.1	32.14	....	....	32.43	32.37	32.35	31.24	31.24	35.10
6. SiO_2 ..... do	10.79	9.19	17.70	18.37	10.0	12.9	9.5	7.96	10.98	20.42	16.60	14.89	19.97	17.41	16.48	14.84
7. Spiegeleisen ..... tons per day	100.5	134.0	91.7	85.5	107.0	88.0	81.3	69.0	153.5	60.0	117.3	127.0	71.4	50.6	59.3	83.8
8. Slag ..... pounds per ton	2,150	1,572	3,450	2,830	2,600	2,500	2,100	2,450	1,810	3,960	4,835	4,060	4,585	4,600	4,735	3,109
9. Mn in alloy ..... per cent	18.7	20.2	16.7	19.9	17.0	19.2	18.0	16.1	19.3	17.8	17.8	17.0	15.9	16.7	16.1	17.48
10. Si in alloy ..... do	1.0	2.3	2.8	2.5	0.7	0.5	10.8	91.6	....	1.9	0.8	0.9	2.9	2.5	4.1	1.77
11. Blast temperature ..... °F	1,283	1,198	1,275	1,088	1,608	1,316	1,172	942	....	....	880	755	768	690	909	1,080
12. Top temperature ..... °F	500	....	890	490	505	750	804	942	....	....	511	522	535	593	483	572
13. Air per minute ..... cubic feet	19,300	20,500	18,700	19,500	....	....	....	....	....	....	30,000	30,300	28,800	25,000	21,300	....
Slag analysis:																
14. CaO ..... per cent	23.9	27.1	30.2	27.7	40.2	49.7	53.5	39.8	43.7	46.4	49.0	50.8	44.0	46.7	46.1	42.18
15. MgO ..... do	11.0	16.2	11.1	9.0												
16. Al_2O_3 ..... do	15.7	15.8	9.5	8.3	13.9	12.2	12.2	14.8	16.4	11.1	9.4	9.8	9.7	11.0	11.3	12.41
17. SiO_2 ..... do	30.8	30.9	30.8	33.9	34.8	30.9	31.2	30.2	30.5	34.5	35.1	31.1	36.5	35.4	36.3	33.48
18. Mn ..... do	10.24	6.35	2.79	6.50	7.37	4.36	11.71	10.58	6.02	4.93	3.83	2.85	6.38	4.14	3.66	7.56
19. Carbon ..... pounds	2,720	2,620	4,398	3,780	2,800	2,796	2,864	3,735	2,870	3,630	4,370	4,195	4,790	5,820	4,110	3,444
20. Mn charged ..... do	739	667	516	593	912	615	570	903	564	587	571	576	563	574	486	671
21. Mn to alloy ..... per cent	66.7	67.4	72.5	75.0	40.6	69.8	70.7	41.1	76.7	68.0	69.8	66.0	63.4	65.2	73.6	60.22
22. Mn to slag ..... do	29.9	15.0	18.6	31.0	20.2	17.7	39.3	28.8	19.3	33.5	32.4	20.2	51.8	33.2	35.6	32.75
23. Mn lost in stack ..... do	13.4	17.6	8.9	—6.0	39.2	12.5	—10.0	30.1	4.0	—1.5	—2.2	13.8	—15.2	1.6	—19.2	6.72

DISCUSSION OF DATA ON FERROMANGANESE PRACTICE.

GENERAL DESCRIPTION OF FERROMANGANESE PRODUCTION.

No specific information on the production of ferromanganese in the blast furnace is available in technical literature. Certain principles are generally understood to be true; these probably being: (*a*) The blast temperature should be high, (*b*) the slag should be basic, (*c*) the hearth temperature should be high.

HIGH BLAST TEMPERATURE.

Table 17 shows that temperatures as low as 735° F. have been used. The temperature for all 40 runs was only 1,135° F. In 1876, ferromanganese was made at the Diamond furnace, Cartersville, Ga.[a] A cold blast driven by a water wheel through a single 3-inch tuyère was employed. In making a ton of 55 per cent ferromanganese, 6,050 pounds of siliceous native ore, with 35 per cent Mn, was used, indicating a 58 per cent recovery. Comparing this with period 22, furnace J, Table 12, where a manganese recovery of only 55 per cent was made with 1,100° F. blast temperature and a 35 per cent ore, it will be seen that whatever economy results from the hot blast, it has not been found at all necessary to actual and profitable operation.

BASIC SLAG.

The average ferromanganese slag represented in Table 17 has the composition 41.7 per cent base, 14 per cent alumina, 28.1 per cent silica, and 10.6 per cent manganese. The average pig-iron slag[b] runs 48.7 per cent base, 13.1 alumina, and 35.3 silica. If a "more basic slag" means that the percentage of base is greater, then the ferro slag is 7 per cent less basic than the pig-iron slag. This method of comparison, however, is hardly proper. The function of the ferro furnace is to reduce the MnO from the slag and leave a practically irreducible calcium and magnesium aluminosilicate. This reduction is never complete. What actually exists may be considered to be reducible MnO as a solute dissolved in the quarternary mixture of irreducible oxides (CaO, MnO, Al_2O_3, SiO_2) as a solvent. It is the composition of the solvent that will affect the extent of the MnO reduction reaction. Compared on the basis of base + alumina + silica = 100 per cent, the two slags analyze as follows:

Analysis of pig-iron slag and of ferromanganese slag.

		Pig iron.	Ferromanganese.
Bases	per cent..	50.3	49.8
Alumina	do....	13.4	16.7
Silica	do....	35.3	33.5

---

[a] Ward, W. P., Manufacture of ferromanganese in the blast furnace: Trans. Am. Inst. Min. Eng., vol. 5, February, 1876, p. 611.

[b] Feild, A. L., and Royster, P. H., Slag viscosity tables for blast-furnace work: Tech. Paper 187, Bureau of Mines, 1918, p. 17.

The percentage of bases is identical for the two slags. The only difference is that the ferromanganese slag carries a little more alumina and a little less SiO_2. This difference, moreover, is due to the character of the materials used and is not the result of any metallurgical design.

HIGH HEARTH TEMPERATURE.

At furnaces A, B, F, I, J, and K, temperatures were taken by means of several Leeds & Northrup optical pyrometers, with telescope sighted (1) through the tuyères both with and without a tuyère glass interposed; (2) on the surface of the slag at flush; and (3) on the surface of the metal at cast. As was to be expected, the temperatures observed opposite the tuyères varied over a rather wide range. To indicate the normal temperature variation in practice, the following comparison was made: Two hundred individual readings taken at seven furnaces, chosen to give a minimum, averaged 1,438° C. These measurements at each furnace were read by at least two observers, with either two or three separate pyrometer sets, through two or more tuyères, and at five well separated periods of time scattered through two to five days. The same sort of an average selected to give a maximum read 1,653° C. The difference between the maximum and minimum here can not well be attributed to any unusual furnace operation or to errors of observations or to instrumental errors. In the face of these figures, it must be accepted that as furnaces are run at present, a day-to-day variation of 225° C. in hearth temperature is not abnormal.

Laboratory tests strongly indicate that 1,265° C. is the lowest temperature at which MnO is reducible by either carbon or carbon monoxide. The lowest practical hearth temperature therefore may safely be set at 1,350° C. Superpose a variation of 225° C. on this minimum and the limits read 1,350° and 1,575° C. The maximum tuyère readings at the hottest furnace observed averaged 1,750° C. The minimum readings at the coldest furnace observed averaged 1,375° C. Where a temperature phenomenon such as this covers almost the whole range of practicable metallurgical temperatures, there is no room left on the temperature scale which the ferro furnace can monopolize as exclusively its own. Average temperatures taken on these seven furnaces were approximately as follows, the tuyère readings being corrected for the absorption by the tuyère sight glass when present and the temperature of the metal and slag corrected for emissivity as given by Burgess:[a]

	°C.
Temperature opposite tuyères	1,550
Temperature of metal at cast	1,386
Temperature of slag at flush	1,426

---

[a] Burgess, G. K.: Temperature measurements in Bessemer and open-hearth practice: Bureau of Standards Technologic Paper 91, 1917, pp. 8-9.

RÉSUMÉ OF CONDITIONS IN FERROMANGANESE FURNACE.

It has been shown that in changing a furnace over from iron to ferromanganese it is neither necessary nor usual to raise the blast temperature, to increase the basicity of the slag, or to run with a hotter hearth.

True, there are certain resultant changes in the practice, the most important one being that the coke requirements are tripled. The daily tonnage is reduced to about one-third; the top temperature rises to about 1,000° to 1,500° F.; the carbon monoxide content of the top gas is increased about 50 per cent; and the blast pressure drops to 3 to 8 pounds per square inch. The operation of the furnace becomes easier; stock descends without hanging, slips become infrequent, fewer tuyères burn out, sulphur is taken care of completely as MnS in the slag, and further, there are no silicon requirements to be met. All furnace difficulties practically resolve themselves into one problem—to get into the metal the maximum percentage of the manganese charged.

THEORY AS TO FUEL REQUIREMENTS FOR PRODUCING FERROMANGANESE.

A number of heat balances with values in B. t. u. have been published on the iron furnace, but it seems hardly justifiable to construct a heat balance for the ferro furnace for comparison, for the following reasons: At room temperature, the heat of combustion of $C+O_2$ to CO_2 is either 4,250 or 4,430 B. t. u. per pound of C; at 1,500° C. it is either 4,550 or 4,950 B. t. u., depending on the authority selected. There is a wide lack of agreement among the few existing determinations of the heat of combustion, and greater lack of agreement among the experimental values for the specific heat of carbon, of oxygen, and of CO. At 600° C. the specific heat of carbon may be 0.31 or 0.44. At metallurgically interesting temperatures (between 1,400° and 1,800° C.), the discrepancy in figures for the total heat of carbon is about 30 per cent.

The specific heat of CO and of N_2 at 1,600° C. may be either 0.266 or 0.298. For the iron furnace the heat imparted to the gaseous products of reaction per pound of metal is either 972 or 1,280 B. t. u. The rise in temperature of gases above that of the burning coke is, therefore, either 696° or 1,070° F., 54 per cent discrepancy. The continued reprinting of some arbitrarily selected figure for the necessary thermal constants is both practically and scientifically objectionable and tends almost invariably to erroneous conclusions.

It is possible, however, to give an idea of the reason for the difference between the fuel requirements of the iron and those of the ferromanganese furnace and yet avoid excessive accumulation of meaningless B. t. u. figures. The comparison following is of interest.

Comparative operating data for pig iron furnace and for ferromanganese furnace.

	Pig iron.	Ferromanganese.
Carbon charged, pounds	1,727	5,524
Stone charged, pounds	984	2,349
Slag, pounds per ton	1,160	3,196
Gas analysis:		
CO_2 (by weight), per cent	21.43	10.44
CO, per cent	23.13	31.00
N, per cent	55.02	58.36
Blast temperature, °C	555	613
Top temperature, °C	204	363

The figures for the iron furnace are the averages for eight of the largest blast furnace plants in the country. The figures for the ferro furnace are the averages from Table 17. The calculations of the weight of the blast and of the gas and the determination of the amount of carbon absorbed by CO_2 and that burned at the tuyères for the ferro furnace are given in Table 19 following.

TABLE 19.—*Selected data on operation of ferromanganese blast furnaces.*

	Pounds.
Carbon charged per ton of alloy	5,523
Carbon combined with metal (6.5 per cent C)	145
Carbon blown out of stack as dust (9.6 per cent assumed)	530
Carbon available as fuel	4,848
Carbon from CO_2 in stone	263
Total carbon gasified	5,111
Carbon in top gas by weight, per cent	0.1615
Top gas weight (per ton of alloy: 5,111÷0.1615)	31,647
Weight of N_2 in top gas: (0.5836×31,647)	18,469
Weight of air blast: (18,469÷0.7672)	24,073
Weight of O_2 in blast: (24,073×0.2310)	5,561
Weight of carbon burned at tuyères: (5,561÷1.333)	4,172
Carbon absorbed in stack (4,848−4,172)	576

The two most important thermal reactions in the blast furnace are:

(1) $C+O=CO+4,450$ (or 4,950) B. t. u. per pound of C (at 1,600° C.).

(2) $CO_2+C=2CO+5,410$ (or 5,900) B. t. u. per pound of C (at 850° to 1,300° C.).

In reaction 1 both the carbon and the oxygen are supposed to be at the temperature of the solid stock in the combustion zone. As already shown this temperature is probably 1,550° C. for the ferro furnace and 1,590° C. for the iron furnace. Subtracting from the total heat of the reaction sufficient heat to bring the blast up to the temperature of the solid stock in the combustion zone gives a net heat to the gaseous product of reaction of 972 (or 1,280) B. t. u. per pound of metal made for the iron furnace and of 3,690 (or 4,800) B. t. u. for the ferro furnace.

In the ferro furnace, either in the hearth, at the tuyères, or just above them all the manganese in the metal was reduced from MnO. All of the CO_2 resulting from this reduction is itself reduced by solid carbon to CO; there is involved in this reaction 0.218 pound of carbon per pound of metal, resulting in a loss of 1,180 (or 1,290) B. t. u., leaving 2,500 to 3,500 B. t. u. to be used in heating the ascending gas above the temperature of the solid stock. The ferro-furnace gas weighs 12.6 pounds per pound of metal and the iron-furnace gas 4.2 pounds. Hence, in the ferro furnace the amount of heat available for raising the temperature of the gases is two to three times greater than in the iron furnace, and the weight of gas to be heated is three times as great. The rise in temperature, therefore, is either the same or less in the ferro furnace than in the iron furnace.

It is unwise to go here into any detailed consideration of why it is necessary to heat the ascending gas to this rather definite but somewhat unknown elevation above the temperature of the descending stock. It may be pointed out, however, (*a*) that some 250 tons of solid stock must be heated from 70° F. to above 2,800° F. in 24 hours; (*b*) that the greater part of this stock, being coke and mineral oxides, has a low thermal conductivity; (*c*) that the ascending gas stream does not have a chance to flow around every separate piece of coke, stone, and ore; and (*d*) that the transfer of heat from gas to solid is, under the most favorable conditions, a slow process. In order to heat the stock in the required time, therefore, the gases must at every point in the furnace be appreciably hotter than the stock; and this means not several degrees hotter but several hundred degrees hotter. Thus, the 4,165 pounds of carbon burned at the tuyères in the ferro furnace gives the gas stream no greater temperature elevation above the solid stock than does the 1,386 pounds burned in the iron furnace.

COMPOSITION OF FURNACE GAS.

At five furnaces samples of downcomer gas were taken and analyses made for the CO and CO_2 contents. The results obtained were somewhat variable but indicated a distinct tendency toward what might be called a normal composition.

As regards oxygen the evidence was that the true furnace gas contains no oxygen. Well taken samples showed 0.0 to 0.2 per cent. It was felt, however, that this oxygen could be safely attributed to leakage of air into the sample. Where the conditions of sampling permit the chance of contamination by the atmosphere, the oxygen content might be as high as 1.0 per cent.

The CO_2 content varied between the extreme limits of 11.6 and 3.5 per cent, with averages for the five furnaces of 6.7, 7.6, 7.1, 5.9, and 6.0 per cent. The CO content varied from 25.0 to 38.0 per cent, with averages for the five furnaces of 31.8, 34.2, 30.9, 30.0, and 31.0 per cent. The average analysis for the five furnaces was CO_2, 6.67 per cent; CO, 31.7 per cent.

Moissan[a] records an analysis of gas from a furnace making 60 per cent ferromanganese as CO_2, 5.5 per cent; CO, 30.0 per cent. This run, made in 1876, was one of the earliest attempts at the production of manganese in blast furnaces.

The analyses of the gas from nine of the largest iron blast furnace plants in the country show the following CO and CO_2 percentages:

Percentages of CO and of CO_2 in gas from nine iron blast furnace plants.

	CO_2	CO.
Furnace 1	12. 6	26. 0
Furnace 2	13. 2	25. 4
Furnace 3	13. 0	24. 3
Furnace 4	16. 3	22. 8
Furnace 5	14. 3	24. 3
Furnace 6	14. 9	23. 5
Furnace 7	15. 6	23. 2
Furnace 8	14. 6	24. 9
Furnace 9	13. 3	25. 4

The average complete analysis of the nine samples is:

Average analysis of gas from nine iron blast furnaces.

	Per cent by volume.	Per cent by weight.
CO_2	14. 34	21. 43
CO	24. 32	23. 13
CH_4	. 36	. 20
H_2	3. 16	.22
N_2	57. 82	55. 02

Assuming from lack of better information that the H_2 and CH_4 content of the gases of the ferro furnace is the same as that of the iron furnace, and estimating the N_2 by difference, the analysis by volume and by weight of ferro gas is:

Assumed average analysis of gas from nine ferro furnaces.

	Per cent by volume	Per cent by weight.
CO_2	6. 67	10. 44
CO	31. 17	31. 00
CH_4	.36	.17
H_2	31.6	. 21
N_2	58. 64	58. 18

"DIRECT" AND "INDIRECT" REDUCTION.

In Table 19, the weight of the furnace gas is calculated to be 31,647 pounds per ton of metal, which means, from the above analyses, 3,300 pounds of CO_2. The CO_2 from 2,341 pounds of the stone is 968 pounds, from the reduction of 190 pounds of Fe_2O_3, 156 pounds, and from the reduction of 3,770 pounds of MnO_2 to MnO by CO, 1,885 pounds; total, 3,009 pounds. This indicates a 9.56 per cent CO_2 content in the gas as against 10.44 per cent given on page 121. There is left 291 pounds of CO_2, which may be due to reduction of MnO by CO in the lower part of the furnace, to carbon deposition from CO in the upper part of the furnace, or—more likely—to experimental error either in the analyses, or in their application to the average practice given in Table 17. The evidence points closely to a 100 per cent reduction of MnO "directly" by carbon.

---

[a] Moissan, H. H., Le manganese et ses composes, Encl. Chimique, 1886, p. 26.

It is well to point out that "direct" reduction by carbon probably means nothing of the kind. As in most important metallurgical problems, no experiment yet made settles this question. It can be assumed that the reaction:

$$MnO + CO = Mn + CO_2 \qquad (A)$$

is the reaction by which the final reduction of MnO takes place at a temperature somewhere between 1,265° and, say, 1,400° C. At this temperature the following reaction takes place, almost to completion, and with great rapidity:

$$CO_2 + C = 2CO \qquad (B)$$

Thus the result of the two reactions going on simultaneously, although not necessarily in the same part of the furnace, gives the same products as the single reaction:

$$MnO + C = Mn + CO \qquad (C)$$

As A and B together are not distinguishable from C by any calculation based upon gas analysis, one must be content to realize that either of these two reactions may be the one that takes place.

BURDENING AND DRIVING THE FURNACE.

It is all very well to apply to the furnace records accepted metallurgical theory, but practically the important thing is to have at hand something definite in the way of a guide to burdening and driving the furnace. The following discussion, therefore, is based exclusively on the figures in Table 17 and is free from any theoretical considerations whatever. It is an attempt to establish from the whole mass of data certain relationships between the 23 items of each run. There are only three factors that the furnace operators can change at will. They are (1) ratio of coke and ore; (2) ratio of stone and ore; and (3) quantity of wind blown per minute. It is assumed here that the blast temperature is kept as high as stove conditions permit. The number of rounds charged and the tons made per day are decided by the furnace itself. With the above three factors determined, there is nothing left for the operator to do but watch the furnace.

FUEL REQUIREMENTS.

Aside from the relatively constant amount of fuel needed to preheat, reduce, and melt the metal, the amount of carbon fuel required per ton of alloy should depend upon the temperature and humidity of the blast, the weight of the slag, and the temperature maintained in the combustion zone and hearth. That is, there should exist a relationship between five variables. In an attempt to determine this relationship from the data taken, the question of humidity must be dismissed from consideration, not necessarily because it is an unimportant factor but because no information is available concerning it. In the matter of hearth temperatures, no direct measurements are at hand; the pyrometer observations made by the bureau's investigators covered too short a period to prove of material assistance at this stage of the investigation. In a general sense the silicon in the alloy may be regarded as a thermometer, but for a given hearth temperature the amount of silica reduced to silicon undoubtedly depends on a number of other factors, including probably the composition of the slag and the speed of operation. The best indication of the temperature of the hearth is given by the amount of unreduced MnO in the slag.

The following procedure was adopted: The results of the 40 runs were arranged in order of decreasing slag weight and placed in four groups. The averages by groups appear in the first part of Table 20. The results were also arranged in the order of decreasing blast temperature, being again placed in four groups and the results averaged by groups. The averages are given in the second part of Table 20 following.

TABLE 20.—*Results of furnace runs arranged to show variation of carbon fuel with weight of slag and with blast temperature.*

VARIATION WITH WEIGHT OF SLAG.

Slag per ton.	Carbon per ton.	Blast temperature.	Carbon reduced to blast temperature of 1,130° F.
Pounds.	*Pounds.*	*° F.*	*Pounds.*
4,728	6,410	990	6,224
3,334	6,150	1,080	6,083
2,604	4,830	1,210	4,936
2,120	4,550	1,220	4,670

VARIATION WITH BLAST TEMPERATURE.

Blast temperature.	Carbon per ton of alloy.	Slag per ton of alloy.	Carbon reduced to 3,250 pounds of slag.
° F.	*Pounds.*	*Pounds.*	*Pounds.*
1,358	4,670	2,700	5,160
1,210	5,160	3,000	5,380
1,077	5,680	3,400	5,530
862	6,100	3,900	5,820

By the method of successive approximation it was found that 1 pound of extra slag requires 0.88 pound of carbon, and 1° F. extra blast temperature saves 1.33 pounds of carbon. Written in the form of an equation:

$$Ka=4170+0.88\ V-1.33\ T \qquad (1)$$

Where

Ka=pounds of carbon per ton of alloy used by the average furnace,
V =pounds of slag per ton of alloy,
T =blast temperature in degrees F.

This equation shows, for a given slag weight, for a given blast temperature, and for average humidity, the quantity of carbon required to heat the materials in the hearth to the average hearth temperature. As has been shown, this temperature is probably about 1,550° C., but it is better not to place too much confidence in this figure.

It is of course possible to charge either more or less carbon than the amount given by equation *1*. The average hearth temperature used by any given furnace operator means merely that temperature found by experience, or assumed from lack of experience, to be the most desirable to maintain. If the operator decides to use a higher temperature in practice, the furnace can be given "excess carbon," a phrase taken to mean the carbon actually charged minus the amount indicated by equation *1*. When excess carbon is used, the percentage of manganese in the slag will be lower, provided all the other conditions remain constant. On the other hand, the carbon charged may be considerably lower than that given by equation *1*, with a resultant hearth temperature lower than the average.

PERCENTAGE OF MANGANESE IN SLAG.

Determination of the percentage of manganese that goes into the slag is approached in the same manner as the problem of fuel requirements, with the exception that there must not be any "left over" variable such as excess carbon over requirements. The operator can at his own free will charge so little fuel that the hearth freezes, or he can charge so much fuel that he bankrupts the owners. The percentage of manganese in the slag, however, is the result of a number of metallurgical phenomena and is definitely determined by the actual furnace facts.

The data in Table 17 show that the percentage of manganese in the slag depended upon the following factors:

(1) The ratio of bases to silica in the slag;

(2) The hearth temperature as determined by the excess or deficiency of fuel above or below that indicated in equation *1*;

(3) The rate of driving.

RATE OF DRIVING.

The expression found most nearly to represent the speed of operation was the pounds of gross slag produced per minute per square foot of hearth area. Gross slag here means the weight of slag-forming materials, manganese being a slag-forming material; that is, the weight of slag that would be flushed from the furnace if none of the manganese was recovered in the metal. There is an implication here—probably warranted—that the MnO_2 charged is reduced to MnO in the upper stack, but that the MnO entering the bosh is fused with the silica, alumina, and bases, and that it is reduced to metallic manganese only after it has descended molten into the hearth. The calculation of the gross slag is simple. To take the figures from the average of the 40 runs represented in Table 17, as an example, the slag weight is 3,196 pounds. The metallic manganese in the metal is 1,680 pounds. This metallic manganese was reduced from 2,170 pounds of MnO. The gross slag was therefore 3,196 plus 2,170, or 5,366 pounds per ton of metal.

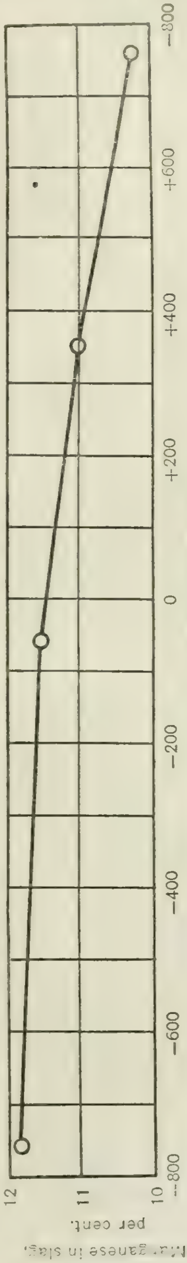

FIGURE 10.—Curve showing relation of percentage of manganese in slag to carbon difference (carbon charged minus carbon required for average hearth-temperature conditions; see Table 17).

It is well to notice that the more usual expression for speed of operation in blast-furnace parlance is the pounds of carbon burned per minute per square foot of hearth area. Without commenting upon this point of view as it applies to the iron furnace, its applicability to the ferromanganese furnace is extremely doubtful. The only factor limiting the speed of operation, assuming that the speed is not so great that the stock is blown out the top of the furnace, is the ability of the hearth to reduce manganese from the slag. As long as the hearth temperature is above 1,265° C., all the manganese in the slag should be reduced, provided the slag remains for a sufficient length of time in the hearth under the reducing influence of a CO atmosphere and in contact with solid carbon. Therefore, no principle of common sense is violated in taking as the criterion of speed the figure for the pounds of slag passing into the hearth from the bosh per minute per square foot of hearth area.

EQUATION FOR PERCENTAGE OF MANGANESE IN SLAG.

The results of the 40 runs represented in Table 18 were arranged in order of decreasing "excess carbon" as determined by equation 1. The results were then divided into four groups and the averages by groups recorded as in Table 21. The results are also plotted in figure 10. The results of the same runs were rearranged in the order of decreasing basicity (ratio of bases to silica), grouped, and averaged, and the averages recorded as in Table 22. The results are also plotted in figure 11. In the same way Table 23 gives the averages for groups taken in order of decreasing rates of driving. The results are also plotted in figure 12. The three tables follow.

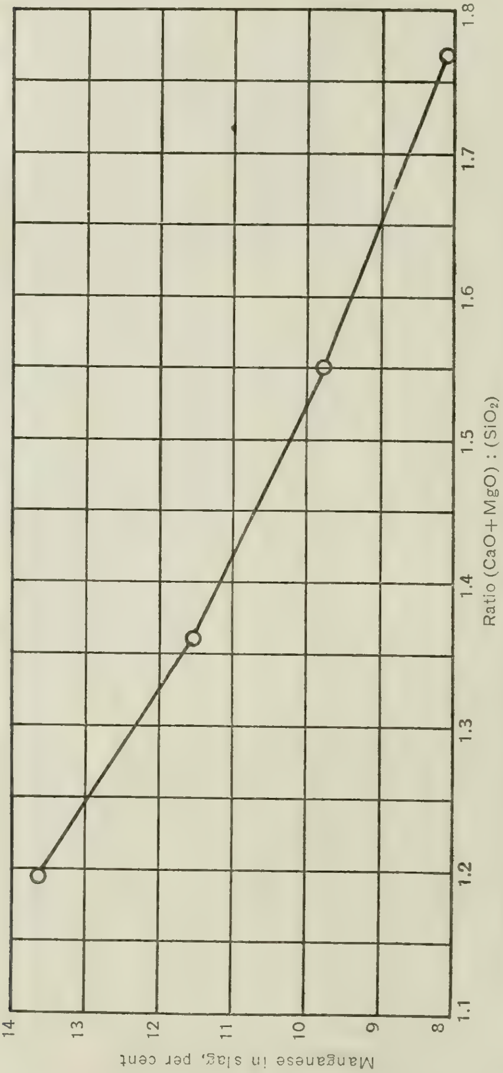

FIGURE 11.—Curve showing relation of percentage of manganese in slag to ratio (CaO+MgO): (SiO$_2$).

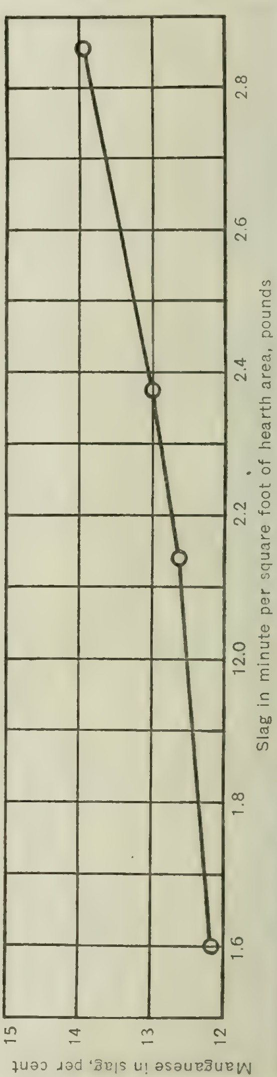

FIGURE 12.—Curve showing relation of percentage of manganese in slag to rate of driving (weight of slag made per minute per square foot of hearth area).

TABLE 21.—*Results of furnace runs arranged to show variation of percentage of manganese in slag with excess or deficiency of carbon fuel as required for average hearth-temperature conditions.*

[Results are plotted in figure 10.]

Group.	Runs included.	Furnaces included.	Carbon excess or deficiency (pounds.)	Basicity ratio.	Rate of driving.	Percentage of manganese in slag.	
						Actual.	Reduced to standard basicity[a] and rate of driving.[b]
1	4, 26, 30, 31, 32, 33, 37, 38, 40	A, C, K, L, M, P	+754	1.69	1.84	8.2	10.3
2	3, 7, 8, 12, 13, 14, 17, 25, 29, 39	A, B, F, K, M	+354	1.48	1.92	11.0	11.0
3	2, 5, 6, 10, 11, 19, 23, 24, 25	A, B, G, J	− 56	1.45	2.18	12.1	11.4
4	1, 15, 18, 20, 22, 27, 34	A, F, G, I, J, K, L	−765	1.37	2.36	13.3	11.8

[a] Standard basicity (ratio of bases to silica), 1.50.

[b] Rate of driving (pounds of gross slag produced per minute per square foot of hearth area), 2.1.

TABLE 22.—*Results of furnace runs arranged to show variation of percentage of manganese in slag with basicity of slag (ratio of $CaO + MgO$ to SiO_2).*

[Results are plotted in figure 11.]

Group.	Runs included.	Furnaces included.	Basicity ratio.	Carbon excess or deficiency (pounds).	Rate of driving.	Percentage of manganese in slag.	
						Actual.	Reduced to standard hearth temperature and rate of driving.
9	6, 8, 23, 24, 25, 26, 28, 31, 33	A, J, K, L	1.77	+411	1.79	7.4	8.2
10	7, 11, 17, 30, 32, 34, 35, 39, 40	A, B, F, L, M	1.55	+187	2.06	9.2	9.9
11	1, 4, 10, 12, 13, 20, 27, 38	A, B, C, I, K	1.36	− 10	2.32	11.8	11.5
12	2, 5, 8, 14, 15, 19, 22	A, B, F, G, J	1.19	−110	2.42	14.2	13.6

TABLE 23.—*Results of furnace runs arranged to show variation of percentage of manganese in slag with rate of driving (pounds of gross slag per minute per square foot of hearth area).*

[Results are plotted in figure 12.]

Group.	Runs included.	Furnaces included.	Basicity ratio.	Rate of driving.	Carbon excess or deficiency (pounds).	Percentage of manganese in slag.	
						Actual.	Reduced to standard basicity and temperature.
5	1, 4, 6, 7, 14, 15, 18, 22, 39	A, B, F, J, M	1.38	2.86	− 60	13.0	13.9
6	2, 5, 8, 19, 20, 21, 25	A, G, I, J	1.70	2.38	−312	12.0	12.9
7	3, 10, 11, 12, 19, 23, 27, 40	A, B, G, J, K, M	1.51	2.15	− 62	12.6	12.5
8	13, 15, 17, 24, 26, 28, 38	B, C, F, J, K	1.64	1.60	+240	10.3	12.2

In deriving an equation for the percentage of manganese in the slag four quantities are involved. If only figures could be selected wherein two of the quantities varied while the other two quantities remained constant, the derivation of the equation would be direct and simple. Unfortunately this is not possible. Thus, in Table 22 there is a distinctly apparent relation between "carbon excess or deficiency," basicity, and rate. As the carbon decreases the basicity decreases and the rate of driving increases. This relationship is due to the human management of the furnace. The increase of basicity is governed by the stone charged; the carbon excess or deficiency by the coke charged; and the rate of driving by the speed of the engines. For any run the stone can be decreased, the coke increased, and the engines slowed down by practically any arbitrary amount, and the furnace will still continue to make ferromanganese, though it may not make a profit. The fact, therefore, that this apparent relationship exists between excess carbon, basicity, and speed, shows only that the furnace operator followed, perhaps unknowingly, a definite rule in burdening and driving the furnaces.

Although in Tables 21, 22, and 23 all the four quantities concerned vary, it is found by successive approximation that—

(*a*) 100 pounds of excess carbon lowers the manganese in the slag 0.11 per cent;

(*b*) An increase in the rate of driving of 1 pound of slag per minute per square foot of hearth area increases the manganese in the slag 1.34 per cent;

(*c*) An increase in the ratio of base to silica of 1 unit decreases the manganese in the slag 9.16 per cent.

In Table 21 values for the actual percentages of manganese in the slag are given in the seventh column. In the last column the values have been corrected for each of the four groups for the amount by which the basicity ratio differs from 1.50, using the correction indicated in (*c*); the values are further corrected for the amount by which the rate of driving differs from 2.1, using the correction indicated in (*b*). In the same way for Tables 22 and 23 the percentages of manganese in the slag are given in the last column corrected for variations of the two quantities, in the fifth and sixth columns, in order to show the relationship that would be found between the variable shown in the fourth column and the manganese in the slag, were the other two variables held constant. These relationships are plotted in figures 9, 10, 11, and 12. Combining these three relations in a single equation gives the result—

$$M = 21.4 - 9.16B + 1.34R - 0.0011(K-K_a) \quad (2)$$

where

M = percentage of manganese in the slag;
B = ration of base to silica;
R = rate of driving (pounds of gross slag per minute per square foot of hearth area);
K_a = carbon required by equation *1*;
K = actual carbon charged.

Combining equation *2* with equation *1* gives:

$$(M=21.4+1.34R-9.16B-0.0011\mathrm{K}-\mathrm{K}_a+0.00097V-0.00145T \qquad (3)$$

Equation *3* is the general solution for *M*, the percentage of manganese in slag, *M* being a function of five variables. The assumption has been made in deriving this equation that the function was linear with respect to all five variables. This certainly is not true. The effect of speed on the reduction of MnO will be, itself, a function of the temperature. Moreover, a large number of variables undoubtedly affecting *M* have been omitted, for example, the physical character of the stock, the temperature of the top gas, the amount of carbon absorbed by CO_2, and the humidity of the blast. It can be hoped only that equation *3* includes the more important variables.

It will be noticed that in joining the points in figures 10, 11, and 12, a curved rather than a straight line is drawn in each figure. This was done with the hope of giving a hint as to the nature of the curve. In each figure a straight line can be drawn which will not miss the experimental point so much as 0.5 per cent. In view of the limitations of the data 0.5 per cent is much inside the limit of experimental error.

BURDENING THE FURNACE.

Observance of the relations in equations *1*, *2*, and *3* may be of material assistance in the burdening and driving of the furnace. The first step in solving this problem is the calculation by equation *1* of the ratio of coke and stone to ore with an assumed basicity ratio. Then by looking at figures 10, 11, and 12, the furnace operator can know at a glance the change in the manganese content of his slag to be expected from adding to, or trimming down, his fuel, or from changing his basicity ratio. No general principles can be given for the solution of this problem as the most important considerations governing the burdening problem are the cost of the materials used, the daily pay roll, and the furnace overhead. Every change in the burdening and driving factors cuts both ways. Slow driving reduces slag loss and runs up labor and overhead charges. Increasing the proportion of lime in the slag within reasonable limits results in lowering the slag loss, but increases coke consumption, and cuts down tonnage, resulting in lower ore cost, higher fuel cost, and increased labor and overhead charges. Using excess coke results in a slight decrease in the percentage of manganese in the slag, but in general increases the slag weight more rapidly, so that the slag loss, being the product of the slag weight and the percentage of manganese in the slag, is increased. About the only general criticism that can be made of the burdening and driving data is that there is on the whole a tendency to use too much fuel and to drive too slowly. The saving of ore resulting from slow driving, while appreciable, causes a decided decrease in profits of the furnace and, although in line with conservation of materials, will not appeal to the operator who is to pay the labor and overhead charges.

It may be stated that the most effective methods of reducing the slag loss of manganese are: (1) The use of low-ash coke and of low-silica stone and (2) carrying as basic a slag as is practicable. In the matter of selecting a low-ash coke it was not possible, perhaps, in the past year for a number of operators to do more than take whatever they could get.

It was universally realized at the furnace that the presence of a large percentage of silica in the coke was playing havoc with the practice; but it is believed that, had the office fully appreciated the loss in profits occasioned by excess silica a considerable improvement in coke could have been effected. Actual purchases of stone, it is certain, have not been made with the care the problem merited. Calcites and dolomites have been used with silica content as high as 7 per cent. Naturally the office did not purchase this stone on the market without a considerable preferential price, but, as is shown later, stone with this silica content should not be used even if obtained free of charge f. o. b. the furnace.

STACK LOSS.

The sum of the manganese in the slag and that in the metal does not equal that charged into the furnace. The percentage of manganese in the charge not accounted for by the manganese carried from the hearth is for convenience called "stack loss." The value of this quantity in Table 17 varies from 0.8 to 32.1 per cent. Its average is 12.8 per cent. As the average loss of manganese in the slag is only 14.7 per cent, the stack loss is as important as the slag loss. It is a quantity, however, that has not thus far proved itself susceptible to analysis or definite explanation. It may be due to: (1) Volatilization of metallic manganese from the hearth, (2) volatilization of manganese oxide in the hearth or stack, (3) manganese oxide carried off mechanically as ore fines by the blast, and (4) a fictitious result due to error in chemical analyses and weights.

To a certain extent in any individual run the apparent stack loss is surely a combined result of these four causes. But for the 40 runs taken as a whole it is difficult to understand how errors in weights and analyses would give a *loss* in 39 cases, instead of giving for one-half of the runs a "*gain* of manganese in the stack." Unless it can be shown that the analyses for manganese in the ore are systematically high or those for manganese in the slag and metal are systematically low, it must be concluded that the errors are sufficiently eliminated in the average of a number of runs.

There is a possibility that the loss as ore fines carried out mechanically by the blast may account for the whole loss.

The operating data from nine iron furnace plants indicate that 6.5 per cent of the iron charged is not accounted for by the iron in the slag and metal. This figure does not inspire much confidence and is given here only for the sake of comparison. As to the loss by volatilization, it is probably not important practically to distinguish between volatilization of metallic manganese and volatilization of manganese oxide. The answer to this problem will appear doubtless, when the relation is found between stack loss and some other operating quantity. If the stack loss is a volatilization phenomenon, it should increase with a rise in hearth temperature. It should also decrease with any increase in the speed of operation, as the longer a given pound of slag or metal remains in the hearth the greater the volatilization. The 40 runs in Table 17 were arranged in the order of decreasing stack loss and divided into four groups. The average for each of the four groups appears in Table 24 following:

TABLE 24.—*Results of furnace runs arranged to show manganese lost in stack as related to other factors.*

Group	1.	2.	3.	4.
Mn lost from stack, per cent	24.4	15.7	9.1	4.0
Top temperature, °F	994	851	995	904
Mn in slag, per cent	8.6	13.5	11.1	10.6
Si in metal, per cent	1.43	.95	1.15	1.26
Ratio of base to silica	1.61	1.47	1.56	1.49
Rate of driving [a]	2.17	2.32	1.95	2.23
Carbon excess or deficiency, pounds	+253	−86	+82	−121

[a] Pounds of gross per minute per square foot of hearth area.

The results presented in the table show a manganese loss for each group of runs, but no systematic variation of stack loss with any other probably related quantity. High hearth temperatures as indicated either by high-silicon metal, excess carbon, basic slag, or low percentage of manganese in the slag appear not to affect the amount of stack loss. There is only one practical conclusion to be formed—the furnace man must admit that the stack loss is an important physical constant and should operate his furnace on that assumption, or he must conclude that the data here presented are insufficient to permit a solution of the problem and obtain for himself more complete and exact furnace records.

SOME TYPICAL CALCULATIONS.

To illustrate the use of calculations suggested in this report, to indicate how many factors are involved in the simplest detail of furnace practice, and to serve as an introduction to the operating problem, a brief outline of the factors involved in the selection of coke and stone follows.

COST FIGURES FOR AVERAGE PRACTICE.

For the sake of comparison a computation of the approximate furnace costs for the average run represented in Table 17, apart from investment charges, may be of interest.

The average charge per ton of metal is 2.67 tons of ore, 3.16 tons of coke, and 1.03 tons of stone. At the arbitrary prices of $40, $10, and $3 per ton, f. o. b. furnace, the materials cost $149.49 per ton of metal made. The following daily costs are assumed: Six hundred dollars for labor, and $175 for the aggregate of superintendence, chemical laboratory, and reserve for relining, repairs, liability insurance, clerical work, demurrage, supplies, etc. Thus there is a total fixed charge of $775 against a daily production of 51.7 tons of ferromanganese, or a charge of $14.75 per ton. Hence the total cost per ton is $164.24. The selling price on 74.9 per cent ferromanganese is taken to be $267.59 ($250 per ton for 70 per cent alloy plus or minus $3.50 per unit of manganese), so that the profit per ton is $103.35, or $5,343 per day.

COKE ASH.

The following analyses of coke are taken from furnace records and although the coke represented is not typical of that used in average practice, it is not abnormal, as millions of tons of such coke have been used annually at certain iron furnaces for years.

Analyses of coke used in certain iron furnaces.

Sample No.	SiO_2.	Al_2O_3.	Fixed carbon.	Volatile matter.	Sulphur.	Moisture.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
1	2.78	2.47	90.95	0.79	0.41	1.35
2	1.80	2.70	91.12	.47	.44	1.52
3	3.06	2.43	91.06	1.18	.80	1.40

Let it be assumed that coke 1 is available for a ferro campaign and that stone of grade "A," as described in the section following ("Selection of Stone"), is also available. Let the ore have the average analysis given in Table 17. Then it is possible to construct a charge sheet for these materials to give approximately the maximum daily profits, subject to the limitation that none of the operating quantities shall lie appreciably beyond what has been tried out as shown by Table 17. The charge will be then as follows:

Analysis of charge.

Material.	Weight, pounds.	SiO_2.		Al_2O_3.		Bases.		Mn.		Fe.		C.	
		Per cent.	Lbs.	Per cent.	Lbs.	Per cent.	Lbs.	Per cent.	Lbs.	Per cent.	Lbs.	Per cent.	Lbs.
Ore	5,310	8.60	457	2.50	133	......	......	40.33	2,240	5.93	325	......	......
Coke	4,000	2.74	109	2.47	98	......	......	......	......	......	......	91.95	3,620
Stone	2,102	.42	10	.13	3	53.0	1,115	......	......	......	......	......	......
Total	......	......	576	......	234	......	1,115	......	2,240	......	325	......	3,620

The manganese balance will be as follows:

Manganese balance.

	Pounds.
Charged	2, 240
Stack loss (12.66 per cent)	[a] 284
Slag loss (8.43 per cent)	[b] 189
To metal (79 per cent)	1, 767
Accounted for	2, 240

The weight of the slag will be 2,167 pounds, the analysis of the slag being as follows:

Analysis of slag.

Constituent.	Per cent.	Pounds.
SiO_2	26. 05	[a] 561
Al_2O_3	10. 85	234
Bases	51. 75	1, 115
MnO	11. 30	241
S		17
Total		2, 167

[a] 151 pounds to silicon in metal.

The carbon as required by equation *1* would be:

$$K=4,170+0.88\times 2,167-1.35\times 1,250=4,405$$

The "deficiency" of carbon charged below that required by equation *1* is therefore 785 (4,405 − 3,620) pounds, only 30 pounds less than was used in the runs in group 1, Table 21. According to equation *2*, this lack of fuel causes the manganese in the slag to increase only 0.87 per cent. Referring to figure 9 it will be seen that the actual curved line falls somewhat below this assumed straight line.

The ratio of bases to silica is 51.75 ÷ 26.05, or 1.99. This basicity is 1.99 − 1.48, or 0.51 higher than that of the average run, and lowers the percentage of manganese in the slag by 4.67 per cent, according to equation *1*. Figure 12 shows that the curved line lies above the straight line. To be conservative, a decrease of only 3.75 per cent is taken, the smallest value figure 12 will justify.

The gross slag is 2,167 + 1.29 × 1,777, or 4,485 pounds. For comparison, the hearth diameter of the furnace is taken to be 10.78 feet, which gives a hearth area of 91.3 square feet, the average hearth area in Table 16 if each furnace be weighted in the average according to the number of runs represented in the table. A driving rate of 2.61 pounds of gross slag per minute per square foot of hearth area is taken. This gives the output as 77.5 tons per day. The rate 2.61 is 0.54 greater than the average rate in Table 17, and, according to equation *2*, causes a rise in the percentage of manganese in the slag of only 0.71 per cent. Within this part of the curve in figure 11 it is difficult to take a greater rise than 0.75 per cent; which figure will be used.

---

[a] Average.

[b] As shown below.

From the above, then, the percentage of manganese in the slag is—

	Per cent.
Average in Table 17	10. 60
Increase due to low fuel	. 87
Increase due to fast driving	. 75
Decrease due to high basicity	3. 75
Manganese in the slag	8. 57

Slag loss (slag weight times percentage of manganese in slag: 2,167×8.75)	189 pounds.
Slag loss in percentage of manganese charged	8.43 per cent.

Summarizing:

Charge:	
Ore, 5,310 pounds, 2.48 tons, at $40	$99. 38
Coke, 4,000 pounds, 2 tons, at $10	20. 00
Stone, 2,102 pounds, 0.94 ton, at $3	2. 82
Cost of materials	122. 20
Daily labor and fixed charges, $775.	
Charge against 1 ton of metal, at 77.5 tons per day	10. 00
Total furnace cost of 1 ton of metal	132. 20
Selling price, $250 for 70 per cent ferro, with $3.50 per unit bonus and penalty.	
Selling price of 1 ton of ferro containing 79 per cent Mn	281. 50
Furnace profit per ton	149. 30
Furnace profit per day, at 77.5 tons per day	11, 570. 76
Furnace profit per day of furnaces represented in Table 2	5, 323. 00
Gain in profits due to improved materials and practice	6, 247. 76

These figures are disturbing, yet it is difficult to see just where they are wrong. The matter is important. If the 11 furnaces represented in Table 17 had operated on this average practice the possible saving would have been $2,061,760.80 per year per furnace, or, for the 11 furnaces, $22,679,368.80. The practical problem would have been to obtain the high-grade coke and stone involved.

An interesting point in the situation is that the iron furnace using the coke best adapted for the production of ferromanganese, namely, that designated as sample 2 in the table on page 112, used stone with 3.36 per cent SiO_2 (to increase the slag volume), and yet made metal of the following composition: Si, 1.52 per cent; S, 0.049 per cent; P, 0.173 per cent.

With the phosphorus content indicated, the metal is neither foundry nor Bessemer grade. As basic pig, 1.52 per cent silicon is objectionably high, unless a better sulphur content than 0.049 per cent can be attained. The answer is that the coke used was lower in SiO_2 than was really desirable. A good coke consumption and a fair tonnage can be shown with low-silica coke and stone, but in basic practice, where the logical function of the blast furnace is to keep down the sulphur and the silicon, the slag volume should not be so low that it can not take care of the sulphur.

Undoubtedly the operator of the basic furnace above mentioned realized this. Enough commercial pressure would have made him release this coke. If the ferro furnace had paid $20 per ton for low-silica coke, the daily profit would still have been $10,000 per furnace, an improvement of $4,523 per day, or roughly 100 per cent. If the desired quality of coke could have been procured for $20 per ton, and if the other assumptions are correct, the increased profit for any one furnace would have been $1,800,000 per annum.

SELECTION OF STONE.

As a typical problem relating to the selection of stone for use in furnaces producing ferromanganese, two grades of stone may be assumed to be under consideration. One may be designated grade A, containing 53 per cent basic constituents and 0.42 per cent silica; the other may be designated grade B, containing 49.5 per cent basic constituents and 7 per cent silica. Stone of grade A has been used in great quantities at one iron furnace plant, and hence is not of purely hypothetical composition. Stone of grade B has been used in producing ferromanganese.

If all the values, except for the grades of stone, be taken as for the average run represented in Table 17, a comparison can be made as follows:

Comparative results obtained with two grades of stone.

Constituent of charge.	Stone A.		Stone B.	
	Quantity per ton of alloy.	Cost.	Quantity per ton of alloy.	Cost.
	Tons.		*Tons.*	
Ore	2.64	$105.56	2.72	$108.80
Coke	3.12	31.15	3.54	35.45
Stone	1.05	3.15	1.70	5.09
Total	....	139.86	....	149.34
Manganese in metal, per cent	75.6		72.15	
Slag per ton of alloy, pounds	2,890		3,839	

The foregoing data are based on the assumption that the slag has a constant basicity ratio of 1.48 and that an increase of 1 pound in the rate of slag formation requires an increase in carbon of 0.88 pound.

The gross slag produced per ton of metal when stone A is used is 5,150 pounds, and when stone B is used 5,970 pounds. If the rate of driving be assumed to be constant, the tonnages will be 51.7 tons with stone A and 44.65 tons with stone B. The daily fixed charges will be roughly the same for both. There will, therefore, be a charge of $14.90 against a ton of metal if stone A be used and a charge of $17.35 if stone B be used.

The total costs are $154.61 and $166.66, respectively, per ton of alloy. The selling prices are $269.60 and $257.53; the profit per ton, therefore, when stone A is used is $114.99, and when stone B is used, $90.87. The profits per day are $5,950 and $4,060. Thus, the daily loss in furnace profits when stone B is used is $1,890. This loss must be charged to the 75 tons of B stone used per day. Thus, even if this grade of stone were obtained free of charge, the furnace loss from its use would be $22.20 per ton. Therefore, if stone A could be purchased at some premium less than $22.20 per ton, a furnace having no choice other than buying A or B would profit by the purchase of A. Such computations as these could be widely extended showing the actual premiums and penalties for coke and ore as based on the cost to the furnace, but the above example will suffice here to substantiate the following conclusions:

(1) The blast-furnace superintendent should know at all times the cost and the analyses of all materials purchased, and should also be familiar with all purchasable materials even including $40 coke and $20 stone.

(2) The decision as to the purchase of raw material should be left to the furnace superintendent, who should be in complete charge of the operation of the furnace.

(3) Far greater saving in furnace profits can be attained from judicious selection of raw materials than from any changes in operating conditions.

When the market price of the alloy falls, as it doubtless will, and the costs of stone, coke, ore, and labor, readjust themselves, it is only by a continued revision of some such calculations as the above, covering the coke and stone, that the furnace operator can hope to make the best showing.

SUMMARY OF OBSERVATIONS ON FERROMANGANESE DATA.

Operating data were collected from the furnace records of 11 blast furnaces making ferromanganese. These data are presented in Table 17, together with a maganese balance showing the distribution of the manganese charged between metal, slag, and top gas. The table includes 40 "experimental periods" of 10 days each and covers the extreme range of operating conditions for each furnace.

Several thousand pyrometer measurements were made of the temperature in the furnaces opposite the tuyères; of the slag at flush and cast; and of the metal at cast. It is shown that, in general, the ferromanganese furnace operates with a colder hearth and produces colder metal and slag than does the iron furnace.

The results of gas analyses made by the bureau show that practically all of the MnO is reduced "directly" by carbon. This reduction probably takes place in the hearth.

The metallurgical questions involved in the smelting of manganese are briefly discussed. It is shown that although more than three times as much carbon is burned at the tuyères in the ferro furnace as in the iron furnace, the gaseous products from the combustion zone are probably at a lower temperature.

The two losses of manganese in the furnace—the slag loss and the "volatilization" loss—are discussed. It is shown that the percentage of manganese in the slag is decreased by raising the blast temperature, by increasing the basicity of the slag, and by charging more coke; and that it is increased by fast driving, and by carrying a greater slag "volume." A general equation is given for the percentage of manganese in the slag as a function of these five quantities, by means of which the slag loss can be calculated in advance from the charge-sheet data.

The "volatilization" loss is shown not to be affected by the basicity of the slag, the silicon in the metal, the carbon charged as fuel, the rate of driving, or the temperature of the top gas. The results, however, are not convincing when examined carefully, and additional and more exact figures are needed. It is shown that by making five changes in practice the 11 furnaces examined could have raised their recovery to 79 per cent, using the same ore, could have increased their tonnage 59 per cent, and could have increased their annual net profits by more than twenty million dollars. The changes are as follows:

(*a*) Using a better grade of coke existing in sufficient quantities and probably purchasable.

(*b*) Using a better grade of stone.

(*c*) Using less fuel.

(*d*) Running with a more basic slag.

(*e*) Driving faster.

Certain furnaces are shown to have lost $10 to $20 a ton by using high-silica stone.

It is concluded that there is greater room for improvement in the selection of coke and stone than in furnace practice.

DISCUSSION OF DATA ON SPIEGELEISEN PRACTICE.

GENERAL DESCRIPTION OF SPIEGELEISEN PRODUCTION.

The simultaneous smelting of iron and of manganese ores—that is, the production of spiegeleisen—is metallurgically an uneconomical process. The reason can readily be appreciated. The iron furnace and the manganese furnace present problems that, though somewhat similar, are at many points fundamentally different. The most economical operation for making iron is not the most economical for making manganese. It is desirable, therefore, that the two processes be kept separate. The only logical reason, of course, for making spiegeleisen at all is to utilize manganiferous iron ores. Spiegeleisen has not, however, always been made in this way. The manganese content of the metal made is frequently reduced by the addition of iron ore or of iron scrap to the charge. During the year 1918, pressure from several sources was brought to bear upon the manganese producers to lower the manganese content of their alloys. The prevailing standard for ferromanganese was lowered from 80 to 72 per cent, and for spiegeleisen from 20 to 16 per cent. To the extent that this was done with the use of lower grade ores, it served a real purpose by creating a market for otherwise unsalable ores. It should be remembered, however, that such a process was wasteful of coke and manganese, and caused decreased furnace tonnage. The rather analagous method of "sweetening" a spiegel ore mixture with high-grade manganese ore is equally objectionable from the standpoint of a metallurgist.

The average spiegel analysis given in Table 18 (p. 116) shows that in order to produce 1 ton of manganese, 5.78 tons of spiegel was required. This alloy contained about 4.4 tons of metallic iron, the equivalent of 4.64 tons of pig iron. The carbon fuel required to make 5.78 tons of spiegel is 19,900 pounds (5.78 × 3444). Crediting this metal with the amount of carbon required for making 4.64 tons of pig iron, namely, 1,727 pounds of carbon multiplied by 4.64 tons of pig iron, or 8,000 pounds total, it is seen that the ton of metallic manganese in the form of spiegel requires 11,900 pounds of carbon.

It is clear that in 1.33 tons of 74.9 per cent alloy there is 1 ton of metallic manganese and 0.23 ton of metallic iron, the equivalent of 0.25 ton of pig iron. The fuel consumption was 7,100 pounds of carbon (1.33 × 5323) according to Table 17. Giving credit for 420 pounds (1727 × 0.25) for the iron made, 1 ton of manganese in the form of ferro requires 6,680 pounds of carbon. Thus, it appears that a ton of manganese in the form of spiegel requires 78 per cent more carbon fuel than a ton of manganese in the form of ferro.

This result is by no means due wholly, or even largely, to the mere combination of the two smelting processes. Most of the excess fuel requirement is due to the character of the ferro ores and of the spiegel ores used. The manganese and silica contents of the two ores are:

	Spiegel.	Ferro.
Manganese, per cent	12.75	40.33
Silica, per cent	14.84	8.60
Silica per pound of manganese, pounds	1.16	.214

The average spiegel ores were therefore 5½ times as siliceous as the ferro ores.

SLAG COMPOSITION.

Table 18 shows that the average spiegel slag is less basic than either the iron slag or the ferro slag. The following figures will permit a comparison of these three average slags calculated on the basis of base + alumina + silica = 100 per cent.

Comparison of three average slags.

	Base.	Alumina.	Silica.	Ratio (Base to SiO_2).
	Per cent.	*Per cent.*	*Per cent.*	
Pig iron	50.3	13.4	35.3	1.42
Spiegel	47.9	14.1	38.0	1.26
Ferro	49.8	16.7	33.5	1.4[illegible]

Table 18 demonstrates that basicity ratios higher than 1.26 have been successfully used in spiegel practice. In fact, the eight runs showing the highest basicity ratio, namely, Nos. 46, 57, 58, 61, 64, 66, and 67, including furnaces B, D, H, J, M, P, and G, show an average ratio of 1.42—exactly that of the slag in the average iron practice quoted above. The average recovery of manganese by the metal for these eight runs was 70 per cent.

It is of interest to compare these figures with those for the eight runs (42, 43, 50, 51, 52, 53, 54, and 60) which had the lowest basicity ratios, the average being 1.08. The average recovery for these runs was 47.3.

HEARTH TEMPERATURE.

At furnaces C, D, E, H, M, and Q, optical pyrometer observations were made of slag, metal, and tuyère temperatures. The same day-to-day variations in temperature were noted. The averages of all temperatures from the six furnaces are given below and the figures from the ferro-furnace observations are repeated for comparison:

Average temperatures in speigel and in ferro furnaces.

	Spiegel.	Ferro.
Temperature opposite tuyères, ° C	1597	1550
Temperature of metal at cast, ° C	1392	1386
Temperature of slag, ° C	1427	1426

Seemingly, both the metal and the slag flow from a spiegel furnace at the same temperature as from the ferro furnace.

COMPOSITION OF FURNACE GAS.

Analyses of spiegel furnace gas were made by the bureau's field party at only four furnaces. The composition of the gas as indicated by the average of all the observations was as follows:

Average composition of speigel furnace gas.

		By volume.	By weight.
CO_2	per cent	7.08	11.05
CO	do	30.68	30.40
N_2	do	58.72	58.08

These figures are not at all satisfactory. The CO content is nearly as high as in ferromanganese-furnace gas. It is difficult to charge this result to ordinary analytical errors. Leakage of air into the sample would lower the CO content; also, the estimation of CO by cuprous chloride absorption when improperly carried out gives a result too low rather than too high. The average analyses for the four furnaces gave 4.62, 5.62, 8.95, and 9.24 per cent for CO_2 content and 24.42, 29.25, 34.20, and 34.85 per cent for the CO content. The extreme limits for CO_2 were 3.9 and 11.1 per cent; and for CO, 23.3 and 35.1 per cent. This range of variation is the same as that encountered in ferromanganese-furnace gas. This fact points strongly to the conclusion that in both of these alloy furnaces, the chemical reactions in the lower part of the furnace are continuously changing. It might almost be said to prove that for a period of time the bosh and hearth reactions are nearly those of the iron furnace, and that following this period as a result of the settling down of the stock, a large amount of manganese oxide is thrown into the hearth, resulting in a rapid reduction of MnO, with a consequent increase in the ratio of CO to CO_2. Therefore, unless more or less continuous samples of furnace gas are taken over a rather long period, the actual analysis might easily misrepresent the true average composition of the furnace gas.

CARBON BURNED AT THE TUYÈRES.

From what has been said in regard to the gas analyses it will be anticipated that a calculation of the amount of carbon burned at the tuyères and of the carbon absorbed by CO_2 (or "the carbon used in direct reduction") will show figures that will have little significance. Table 25 following shows a calculation of the weight of gas, the weight of blast, and the distribution of carbon between the stack and the tuyères. So long as the amount of carbon blown out as flue dust must be guessed at; and until systematic gas analyses are taken covering appreciable periods of time, no satisfactory figures for these quantities can be given. However, the figures 676 for the ferro furnace, 347 for the spiegel furnace, and 169 for the iron furnace undoubtedly indicate, in a general qualitative way, the pounds of carbon per ton of metal absorbed by CO_2.

TABLE 25.—*Selected data on operation of spiegeleisen blast furnaces.*

Carbon charged per ton of alloy	pounds..	3, 444
Carbon combined with metal (5.0 per cent C)	do....	112
Carbon blown out of stack as dust (9.6 per cent assumed)	do....	330
Carbon available as fuel	do....	3, 002
Carbon from CO_2 in stone	do....	260
Total carbon gasified	do....	3, 262
Gas analysis (by weight):		
CO_2	per cent..	11. 05
CO	do....	30. 40
N_2	do....	58. 08
Carbon in top gas	do....	16. 07
Weight of top gas per ton of alloy	pounds..	20, 300
Weight of N_2 in top gas	do....	11, 810
Weight of air blast	do....	15, 280
Weight of O_2 in blast	do....	3, 530
Weight of carbon burned at tuyères	do....	2, 655
Carbon absorbed by CO_2 or involved in direct reduction	do....	347

FUEL REQUIREMENTS.

No relation can be determined between carbon fuel, blast temperature, and slag weight in the spiegel furnace by the method employed for ferromanganese. Table 18 (p. 116) shows that most of the furnaces operating with high-blast temperatures were working on ore mixtures that gave low slag weights, and the furnaces smelting siliceous ores were running with low-blast temperatures. In attempting to determine K_a (the carbon fuel used for average hearth temperatures) as a function of T (the blast temperature) and of V (the slag weight), taking these last as independent variables, it is difficult to distinguish between the effect of changes in blast temperature and of changes in slag weight, as T and V do not vary independently. The method employed below is not so satisfactory as regards accuracy as that used with ferromanganese, but it is simpler and more direct.

The runs in Table 18 were divided into three groups, selected to conform with the following classification:

1. High-blast temperature and small slag weight.
2. High-blast temperature and large slag weight.
3. Low-blast temperature and large slag weight.

The average values of K_a, T, and V for these three groups are given in Table 26 following:

TABLE 26.—*Average values for carbon fuel, blast temperature, and slag weight in the furnace runs represented in Table 18.*

Group.	Runs included.	Furnaces included.	Carbon per ton.	Blast temperature.	Slag weight.
			Lbs.	°*F.*	*Lbs.*
1	43, 46, 48, 49, 55, 57, 62, 64	C, D, H, M, P	2,821	1.221	1,979
2	41, 42, 44, 52, 54, 58, 59, 60, 61, 70	C, E, J, M, R	3.525	1,142	3,349
3	50, 51, 53, 65, 66, 67, 68, 69	E, B, Q, R	4,175	825	4,392

On the assumption that the carbon fuel required for average hearth temperature conditions can be written in the form—

$$K_a = \mathrm{A} + \mathrm{B}T + \mathrm{C}V$$

where A, B, and C are constants, the figures given in Table 26 are sufficient to determine the value of the three constants. Substituting in the above equation three times gives—

$$\begin{aligned} 2{,}821 &= \mathrm{A} + 1{,}221\mathrm{B} + 1{,}979\mathrm{C} \\ 3{,}525 &= \mathrm{A} + 1{,}142\mathrm{B} + 3{,}349\mathrm{C} \\ 4{,}175 &= \mathrm{A} + \phantom{1{,}}825\mathrm{B} + 4{,}392\mathrm{C} \end{aligned}$$

From which it follows that—

$$K_a = 2{,}365 - 0.44T + 0.49V \tag{4}$$

As previously stated, K is the quantity of carbon per ton of metal used for any particular run. K_a is the quantity that would be required if the same thermal conditions were maintained in the furnace as in the average furnace. Confusion has often arisen in discussions of blast-furnace fuel economy from identifying the carbon "consumed in" the furnace with the carbon "required by" the furnace. It is true in iron furnace practice that for a given product there is a minimum carbon consumption below which the furnace will not work, but there is practically no upper limit. In making manganese alloys, however, it may be said that there is neither a maximum nor a minimum limit. The actual amount of carbon, K, charged into any furnace is merely the amount the operator believes necessary to maintain a given ratio of coke to ore. In each run, therefore, K, the actual carbon will differ from K_a, the carbon that would on the average be charged in a furnace working with that particular blast temperature and slag weight. $K—K_a$ is a measure of the thermal condition of the hearth. It should be true that when $K—K_a$ is positive the furnace is running hotter than when $K—K_a$ is negative, and this difference in available fuel should influence the amount of MnO left unreduced in the slag.

PERCENTAGE OF MANGANESE IN THE SLAG.

It has been shown for the ferromanganese furnace that M, the percentage of manganese in the slag, is a function of three quantities, namely: B, the ratio of base to silica; $K—K_a$, the excess carbon; and R, the rate of slag formation per square foot of hearth area. The relation between the quantities was found to be—

$$M = 21.4 - 9.16\mathrm{B} + 1.34R - 0.11(K - K_a)$$

In the same way, as is shown below, for the spiegel furnace.

$$M = 14.3 - 5.31\mathrm{B} + 0.0R - 0.09(K - K_a) \tag{5}$$

This equation is determined as follows: The runs in Table 18, arranged in order of decreasing basicity, are divided into two groups of 15 runs each, group 1 representing runs having the highest basicity and group 2 those having the lowest. The runs are rearranged in order of decreasing $K-K_a$, and two further groups—group 3, representing runs with excess carbon, and group 4, representing runs with deficiency of carbon—are selected. The average values of M B, and $K-K_a$ for these four groups appear in Table 27.

TABLE 27.—*Results of furnace runs arranged to show variation of manganese in slag according to basicity of slag and according to excess or deficiency of carbon fuel.*

Group.	Basicity.	Excess carbon.	Manganese in slag.
		Pounds.	*Per cent.*
1	1.36	123	6.69
2	1.14	137	8.09
3	1.29	292	6.89
4	1.21	331	8.01

From these figures, assuming that the value of M can be expressed in the form given in equation 5, a simple calculation shows: (*a*) That one unit increase in basicity lowers the manganese in the slag 5.31 per cent; (*b*) That 100 pounds of carbon lowers the manganese in the slag 0.09 per cent.

To show that the rate of driving does not affect the manganese content of the slag, the runs can be again rearranged in order of decreasing R (rate of slag formation per square foot of hearth area), giving two further groups—group 5 representing runs with fast driving, and group 6, representing runs with slow driving. The average values of M, B, $K-K_a$, and R for groups 5 and 6 are given in Table 28.

TABLE 28.—*Results of furnace runs arranged to show variation of manganese in slag according to rate of driving.*

	Group 5.	Group 6.
Basicity ratio	1.24	1.23
Excess carbon, pounds	256	208
Rate of driving, expressed in pounds of slag made per minute per square foot of hearth area	2.86	1.48
Mn in slag, per cent	7.81	7.47

Although the rate of driving differs by 1.38 pounds of slag per minute per square foot of hearth area, the difference of 0.34 per cent manganese in the slag is exactly accounted for by the difference in the values of B and of $K-K_a$. This might well have been anticipated. The curve in figure 10, (p. 127), showing the relation of the percentage of manganese in the slag to the rate of driving for the ferromanganese furnace, is convex downward. When two or more pounds of slag per square foot of hearth area is being produced, the time factor is of importance. The lower end of this curve, however, shows a distinct flattening with a driving rate as low as 1.6 pounds of slag per minute. It is a peculiar coincidence that the average rate of slag-formation in both the spiegel practice and in the ferro practice is just 2.1. As regards merely the reduction of MnO, there seems to be no necessity for running the spiegel furnace as slow as the ferro furnace. There is so much more MnO per pound of gross ferro slag to be reduced that the ferro slag should probably enter the hearth only at one-third the rate that is found economical in spiegel practice.

Seemingly, two conclusions may be drawn from these facts:

(1) That overdriving results in a loss of manganese, while underdriving does not result in any corresponding saving of manganese.

(2) That the spiegel furnaces investigated were driven much slower than is shown necessary by any facts presented in this paper.

It is, of course, possible that through other causes furnace difficulties might arise with fast driving, although no evidence of such an effect is at hand.

STACK LOSS.

The "stack loss" is a factor of less importance in the spiegel furnace than in the ferro furnace; the average value is only 6.72 per cent, whereas for ferromanganese it was shown to be 12.8 per cent. The individual values vary from +39.2 to =19.2, a range of 58.4 per cent. The individual values for ferro stack loss varied over a range of 32.9 per cent.

The stack loss shown in Table 18 does not seem to be connected with any of the other operating quantities. The 30 runs were divided in half, giving two groups of 15 runs each; one group included runs with the larger stack losses and the other included runs with the smaller stack losses. The average values of the figures that may have some possible connection are given in Table 29 following:

TABLE 29.—*Results of furnace runs arranged to show comparative effects of large and of small stack losses.*

Item.	Furnaces with large stack loss.	Furnaces with small stack loss.
Stack loss, per cent	15. 5	−2. 22
Silicon in the metal, per cent	1. 92	1. 57
Basicity	1. 25	1. 27
Blast temperature, °F	1, 130	1, 018
Top temperature, °F	633	525
Slag per ton, pounds	2. 930	3, 320
Manganese in slag, per cent	7. 42	7. 69
Carbon burned, pounds	3, 260	3, 665
Recovery, per cent	55. 9	64. 5
Excess carbon, pounds	−35	110

The difference between the stack losses in the two groups is 17.7 per cent. None of the other factors, however, show enough variation to be responsible for such a change in the stack loss. In the first group the silicon content in the metal is higher, as is also the top temperature, while the manganese in the slag is lower. These are changes in the right direction to account for a stack loss, although they can not explain it quantitatively. The fact that the basicity is identical in the two groups is important, as it has been thought by some that "liming up" the slag tends to increase the stack loss.

SUMMARY OF OBSERVATIONS ON SPIEGELEISEN PRACTICE.

The average recovery of manganese in the production of spiegeleisen was found to be 60.22 per cent, the loss of 39.78 per cent being distributed, one-sixth to the stack and five-sixths to the slag.

Although the individual values for stack loss were very irregular, it appears that they were not affected by the basicity of the slag, the temperature of the top gas, nor the carbon fuel charged. There seems to be no way to reduce this anomalous quantity. On the other hand, the evidence does show that the stack loss will not be increased by increasing the basicity of the slag.

The slag loss is decreased by increasing the basicity of the slag. The average basicity of the spiegel slags investigated was $CaO + MgO \div SiO_2 = 1.26$; it seems that this basicity ratio is uneconomically low. The percentage of manganese in the slag is shown to be independent of the rate of slag formation; it is also shown that the spiegel furnaces are operated much more slowly than the manganese oxide reduction demands.

To make a ton of metallic manganese in the form of spiegel requires 78 per cent more carbon fuel than is required to make a ton of metallic manganese in the form of ferro. This greater fuel consumption is due largely to the enormous amount of silica carried in the spiegel ore with the manganese, namely, 1.16 tons of silica to 1 ton of manganese. In this respect the ferro investigation and the spiegel investigation do not overlap. The most siliceous ferro ores used (containing 0.5 ton of SiO_2 per ton of Mn) have a $SiO_2 : Mn$ ratio just equal to that of the least siliceous spiegel ores used. The successful operation of spiegel furnaces on very siliceous ores, however, indicates that the silica specifications on high grade manganese ores have been much too stringent.

Practically all of the conclusions reached in regard to the production of ferromanganese are applicable to spiegel production. In making speigeleisen, the use of better coke and stone has some advantages, but not sufficient to permit such bonuses for low-silica stone and coke as would be permissible in ferro practice; because the silica in the spiegel slag comes largely from the ore gangue, and because manganese in the form of spiegel ore is less expensive than manganese in the form of ferro ore.

Prices for spiegel ores have not been fixed, as for ferro ores, but are subject to individual contract, so a discussion of profits in spiegel operation will not be attempted.

CHAPTER 9.—NATIONAL IMPORTANCE OF ALLOCATING LOW-ASH COKE TO MANGANESE ALLOY FURNACES.

By P. H. Royster.

INTRODUCTORY STATEMENT.

During 1918 an investigation of manganese alloy furnaces with particular reference to the character and quality of the coke being supplied to the furnaces was conducted by the Bureau of Mines. Twelve typical blast furnaces producing ferromanganese and spiegeleisen were examined. These 12 furnaces produced approximately 40 per cent of the total output of manganese alloys in the United States and offer a fair representation of the industry as a whole.

It developed that the poor quality of the coke in general use was responsible for serious waste, both of manganese and of coke. This fact was of vital interest not only to the Bureau of Mines, but also to the Fuel Administration.

FURNACE DATA.

The 12 furnaces studied furnished the following statistics:

Data regarding 12 blast furnaces producing ferromanganese and spiegeleisen.

Item.	Furnaces making ferro-manganese.	Furnaces making spiegeleisen.
Number of furnaces investigated	6	6
Average monthly output of alloys, July and August, tons	9,966	11,736
Average coke per ton of alloy, tons	3.25	2.12
Total coke consumed, tons	32,387	24,880
Average manganese in alloys, tons	73.0	17.5
Metallic manganese in alloys, tons	7,266	2,100
Conversion loss, per cent	29	38
Metallic manganese in ores, tons	10,266	3,380
Average manganese in ores, per cent	40.6	12.8
Total ore consumed, tons	25,276	26,502

IMPORTANCE OF ALLOCATING LOW-ASH COKE TO MANGANESE FURNACES.

A total of about 57,000 tons of coke per month was being consumed by the 12 furnaces under review. This coke was found to contain an average of nearly 14.5 per cent ash, or over 7 per cent silica.

The principal losses of manganese in the manufacture of alloy are in the slag, these losses being roughly proportional to the slag volume. The controlling slag-forming element is silica. Silica in coke is more harmful than silica in ore because it not only increases the slag volume but also diminishes the efficiency of the coke, making more coke necessary, which in turn introduces more silica, thereby again increasing the slag volume and the coke requirement. In fact, if the coke contains as much as 20 per cent ash, the vicious circle may be complete, all further additions of coke serving no purpose whatever. It is obvious that the more nearly this extreme is reached, the more wasteful the process becomes, both as to manganese and as to coke. Conversely, with better coke, less coke is needed and more manganese is saved.

Comparative observations made and recorded demonstrate if the 12 furnaces under review had been supplied with coke containing 8 per cent ash instead of 14.5 per cent, the same monthly tonnage of alloys could have been produced with about 40,000 tons of coke instead of the 57,000 tons actually used. At the same time, the conversion loss of metallic manganese would have been reduced from 29 per cent to 14 per cent for ferro, and from 38 per cent to 29 per cent for spiegel. The net result of this last would have been a saving of 1,826 tons of metallic manganese, equivalent to 4,560 tons of 40 per cent ore in the case of ferro; and a saving of 420 tons of metallic manganese, equivalent to 3,230 tons of 13 per cent ore in the case of spiegel.

Extending these figures, which represent actual conditions as to about 40 per cent of the country's (1918) output of manganese alloys, to cover the entire estimated requirement of such alloys for the year 1919, gives the following results:

1. Estimated requirement for 1919, 280,000 tons metallic manganese of which 20 per cent shall be in the form of spiegel.

	Tons of coke.
2. 56,000÷17 per cent=330,000 tons spiegel×2.12=	699,600
224,000÷72 per cent=312,000 tons ferro×3.25=	1,014,000
Total requirement of 14.5 per cent ash coke	1,713,600
Saving through use of good quality coke, 31 per cent	532,000
Total requirement of 8 per cent ash coke	1,181,600

	Poor coke.	Good coke.	Saving.
3. Spiegel conversion loss, per cent	38	29	......
Tons Mn needed for spiegel	90,000	79,000	11,000
Tons 13 per cent ore needed for spiegel	690,000	606,000	84,000
Ferro conversion loss, per cent	29	14	......
Tons Mn needed for ferro	315,000	262,000	53,000
Tons 40 per cent ore needed for ferro	786,000	655,000	131,000

In short, by allocating low-ash coke during 1919 to the manganese-alloy furnaces, there may be saved over half a million tons of coke; about 131,000 tons of manganese ore of ferro grade; about 84,000 tons of ore of spiegel grade; and a total of about 64,000 tons of metallic manganese.

The effect of such allocation on the production of pig iron must, however, be considered. The pig-iron industry would be deprived of roughly 1,200,000 tons of good coke and receive instead about 1,700,000 tons of relatively poor coke.

In the manufacture of pig iron, poor coke does not cause any increase in loss of metallic iron. The substitution of 14.5 per cent ash coke for 8 per cent ash coke would, however, cause about 10 per cent decrease in production and about 20 per cent increase in the use of coke. The decrease in production is the direct result of having to use an increased amount of coke.

The net effect would be a decrease in the production of pig iron of about 120,000 tons, and an increase in coke consumption of about 240,000 tons. One hundred and twenty thousand tons is approximately one-third of 1 per cent of the total pig-iron output of the country.

However, as the pig-iron industry would receive 500,000 tons more coke than it would be asked to release there would be enough coke left, after supplying the above additional requirement of 240,000 tons, to make up the deficiency in pig iron and still leave a margin of 120,000 tons.

Therefore the net coke supply would be actually increased, while maintaining the pig output at its present level; but about 130,000 tons additional furnace capacity would have to be provided.

The greater part of this, namely, at least 100,000 tons,[a] would be furnished by the manganese-alloy industry, because its furnace requirements would be reduced by this amount, while producing the same tonnage of alloys, on account of having high-grade coke. The small balance must be made up from some other source. As bearing upon this point, it is interesting to note the following statistics:

Data on pig-iron blast furnaces in the United States.

	Number.
In blast, Aug. 31, 1918	371
In blast, Sept. 30, 1918	364
Out of blast, Sept. 30, 1918	74
Total furnaces, Sept. 30, 1918	438
Average yearly capacity per furnace (approximate), tons	100, 000

CONCLUSION.

Without doubt, a large proportion of the furnaces idle (Oct. 1, 1918) are small and poorly equipped. But, in view of the immense national importance of allocating low-ash coke to the manganese-alloy furnaces, it would seem justifiable to exert pressure on the pig-iron industry to make up the resulting slight loss of furnace capacity from among the furnaces now idle.

The figures given herein are based on a grade of coke which is to-day admittedly scarce; on the other hand, the total requirements of the manganese-alloy furnaces are only a small proportion of the total coke output of the country. The best coke available, whatever its grade may be, should be allocated to these furnaces; and if the average ash proves to be somwhat more than 8 per cent, the savings made, while proportionately less than shown above, will still be of the utmost importance.

In making these figures, commercial aspects have been constantly kept in mind. The figures are not theoretical; they represent what it is believed can actually be accomplished.

---

[a] Furnaces capable of making 68,000 tons of spiegel per year could be released for pig production. The estimate of 100,000 tons pig from these furnaces is low. A 100-ton pig furnace will not make 68 tons of spiegel.

CHAPTER 10.—ELECTRIC SMELTING OF DOMESTIC MANGANESE ORES.

By H. W. Gillett and C. E. Williams.

INTRODUCTORY STATEMENT.

The utilization of domestic manganese ores has been an important war-time problem, and its bearing on national self-sufficiency makes it worthy of attention even in times of peace. This paper deals only with the utilization of low-grade domestic ores by electric smelting.

As a rule, blast furnaces can not, in normal times, economically use ores averaging much below 40 per cent Mn and much above 12 per cent SiO_2, although during the war these limits were exceeded. The United States contains large tonnages of ores lower in Mn and higher in SiO_2 which may be amenable to electric smelting. An electric-furnace plant can be constructed more rapidly and can be economically operated in smaller units than a blast furnace; also it can be situated wherever a suitable supply of ore and a suitable water-power development are found near each other. An ore deposit inadequate to supply a modern blast furnace would last a rather large electric furnace for a considerable time. Hence, the smaller electric unit can be so located that the transportation charges are less than if the ore were shipped to a more distant blast furnace.

The writers wish to acknowledge the valuable assistance of Messrs. E. L. Mack and F. C. Ryan, of the Bureau of Mines, and of Lieuts. L. S. Deitz, jr., and J. G. Thompson, Ordnance Department, and Pvt. E. O. Denzler, of the Chemical Warfare Service, who aided in the experimental and analytical work. Grateful acknowledgment is also made to the Chemical Department of Cornell University for the use of its electric-furnace equipment and of its analytical laboratory.

PRESENT DEVELOPMENT OF ELECTRIC SMELTING OF DOMESTIC ORES.

Electric smelting of local ores has been considerably developed. The Southern Manganese Corporation, Anniston, Ala., has eight furnaces, some of them of 3,000 kilowatt size, operating on local ore.[a] The Anaconda Copper Co.[b] will soon be operating five 3,000-

---

[a] Swann, T., (news item): Met. and Chem. Eng., vol. 18, 1918, p. 512.

[b] Anonymous, Electric furnace for manganese ores: West. Elec., vol 73, 1918, p. 218; Chem. Abs., vol 12, 1918, p. 1856.

kilowatt furnaces at Great Falls, Mont., on uncalcined rhodochrosite ore from Butte, Mont., and will probably also use some ore from Philipsburg, Mont. Washed Philipsburg ore is also smelted by the Bilrowe Alloys Co., Tacoma, Wash., in six 350 to 400-kilowatt furnaces, and this firm has also smelted ore from the Olympic Mountains in Washington.

The Noble Electric Steel Co., Heroult, Calif., smelts California ores in two 1,300 to 1,500-kilowatt furnaces. The Pacific Electro Metals Co. makes ferromanganese and silicomanganese in a 3,000-kilowatt furnace at Bay Point, Calif., from California ores.

The Iron Mountain Alloy Co., Utah Junction, Calif., runs a 1,200-kilowatt and an 1,800-kilowatt furnace on Colorado, Utah, and Nevada ores.

The Western Reduction Co., Portland, Oreg., is just starting the production of ferromanganese in a 700-kilowatt furnace.

ELECTRIC SMELTING PRACTICE ON FERROMANGANESE.

FURNACES USED.

Practice in making ferromanganese differs slightly at the various plants, but in general is uniform. Keeney[a] has recently described the Iron Mountain Alloy Co.'s practice. Furnace voltages (electrode to charge) run from 55 volts in the 350-kilowatt size to 85 in the 3,000-kilowatt size. The furnaces are usually three-phase, three-electrode, open-top furnaces, the exceptions being those of the Bilrowe Alloys Co., which are single-phase, and those of the Noble Electric Steel Co., which are of a three-phase, four-electrode type. The general type of furnace used is the rectangular form described by Lyon, Keeney, and Cullen.[b] The Noble furnace is described by Vom Baur.[c]

The furnace lining is usually carbon or magnesite, rarely water-cooled fire brick. The furnaces are charged continuously and tapped every two hours. Slag and metal are tapped together into a settler. A few plants use round electrodes jointed for continuous feed, which should reduce the electrode consumption. However, in most plants the feed is not continuous, and the butts are scrapped. Seemingly, a Scott-connected, two-phase circuit from a three-phase circuit with the conducting bottom of the furnace as the neutral electrode, and two upper electrodes, might reduce the electrode consumption by presenting only two instead of three electrodes to oxidation, while still allowing the construction of a furnace of high kilowatt capacity without the upper electrodes being required to carry too much current.

---

[a] Keeney, R. M., The manufacture of ferro-alloys in the electric furnace. Bull. Am. Inst. Min. Eng., vol. 140, August, 1918, pp. 1321-1373.

[b] Lyon, D. A., Keeney, R. M., and Cullen, J. F. The electric furnace in metallurgical work: Bull. 77, Bureau of Mines, 1914, p. 113.

[c] Rodenhauser, W., Schoenawa, J., and Vom Baur, C. H., Electric furnaces in the iron and steel industry, 1917 ed., p. 365.

RAW MATERIALS EMPLOYED.

The ores of mixtures used at all these plants vary between 39.5 to 43.5 per cent manganese and 15 to 25 per cent silica with low phosphorus, averaging about 40 per cent manganese, 20 per cent silica, 0.20 per cent phosphorus, and less than 4 per cent ferro. The Great Falls, Mont., plant will be an exception, as it will operate largely on Butte rhodochrosite ore which is extremely low in silica and phosphorus.

The plants all flux the ore with limestone, except the plant at Heroult, Calif., which uses a little fluorspar. Also they produce a slag whose average composition is 10 to 12 per cent Mn., 35 to 40 per cent CaO + MgO, about 30 per cent SiO_2, and such small amounts of Al_2O_3 as are carried by the ores. The latter are all low in alumina with the exception of occasional lots smelted at Anniston, Ala.

AVERAGE RESULTS OBTAINED.

The power consumption and the electrode consumption in the various furnaces, brought to the basis of a 3,000-kilowatt furnace, average 5,000 kilowatt-hours and 170 pounds of carbon electrodes per long ton of 80 per cent ferromanganese.

As the ores are low in iron, a little iron ore, or better and more generally used, some steel scrap, is added to supply enough iron to bring the alloy to 80 per cent manganese. The alloys contain 0.20 to 0.40 per cent phosphorus, the former figure being aimed at, but the latter being taken by the trade while the ferro supply is low. The California ores are low in phosphorus; consequently the California alloys meet the lower figure. The silicon content runs under 2 per cent, though it sometimes goes up to 5 per cent in the Anniston product when ores higher than usual in silica are employed.

The Anniston, Anaconda, and Iron Mountain plants use anthracite coal as reducer; the Bay Point and Tacoma plants use coke; and the Heroult plant uses about half coke and half charcoal.

The recovery of manganese averages 75 per cent, being nearer 80 per cent on ores of 15 per cent silica and nearer 70 on those of 25 per cent silica content. From a sixth to a third of the conversion loss is due to volatilization and dusting, and slag losses account for the rest. In the blast furnace operating on low silica ores these losses are exactly reversed, slag accounting for only a sixth to a third and the rest being due to volatilization and dusting.[a]

---

[a] Newton, Edmund, Manganiferous iron ores of the Cuyuna District, Minn., Bull. 5, Minn School of Mines Sta., University of Minn., 1918, p. 71.

EFFECT OF HIGH SILICA IN ORES.

It is seen that the electric smelting plants use ores higher in silica than are standard for blast-furnace practice, but that 25 per cent silica is the present limit in ores used for making ferromanganese.

The trouble that silica causes is reflected in the manganese price scale of the War Industries Board,[a] by which an ore with 40 per cent manganese and 8 per cent silica is worth $40.80 per ton and one with 35 per cent manganese and 35 per cent silica only, $7.45 per ton. The scale does not run below 35 per cent manganese, but if it did an ore with 27 per cent manganese and 35 per cent silica would be worth nothing, the penalty for silica entirely wiping out the value of the manganese even though calculated at the same price per unit as given in the scale for a 35 per cent manganese content.

CONCENTRATION OF ORES.

The Philipsburg and the Butte, Mont., districts contain vast tonnages of manganese ores containing around 25 per cent manganese and 35 per cent silica, to say nothing of the rhodonite ores of Colorado and of the tailings from concentrating ores by washing or screening.[b]

Some concentration of domestic ores is done,[c] but in general concentration, is not very successful[d] and a slight improvement in the method of handling low-grade ores might make it more desirable to smelt the run-of-mine ore than to attempt concentration. It is stated[e] that some ores could be concentrated after fine crushing; also that leaching and electrolytic deposition of the manganese oxide is possible. The product would, however, be fine powders and would be extremely difficult to smelt in the usual fashion, keeping the shaft full of charge, unless first briquetted or sintered. It is not certain that briquetting or sintering could be successfully applied.

REDUCTION OF SILICON.

Even though the usual basic slag be employed when smelting ores high in silica, silicon is reduced and enters the alloy when the slag volume is high. Newton[f] states that this is true in the blast furnace. Swann[g] also indicates it for electric-furnace operation, stating that slags from electric smelting containing 12 to 14 per cent Mn, 40 per cent CaO, and 30 per cent SiO_2 can be smelted to an alloy containing 60 per cent Mn and 19 per cent Si. Had the original smelting been carried on far enough to reduce more of the manganese, considerable silicon would have been reduced also. A high temperature and a large amount of reducing agent are needed to drive the manganese out of the slag, and these conditions favor the reduction of silicon.

---

[a] Chapter 1 of this bulletin.

[b] See Pardee, J. T., Manganese at Butte, Mont.: U. S. Geol. Survey Bull. 690, 1918, pp. 111-130; Harder, E. C., Manganiferous iron ores; U. S. Geol. Survey Bull. 666-ee, 1917, 13 pp.

[c] See Swann, T., place cited; Anonymous, Manganese concentrator at Philipsburg, Mont.: Met. and Chem. Eng., vol. 18, 1918, p. 625.

[d] See Pardee, J. T., work cited, p. 11; Harder, E. C., work cited, p. 13; Newton, E., work cited, p. 57.

[e] News item, Manganese in California: Chem. and Met. Eng., vol. 19, 1918, p. 702.

[f] Newton, E., work cited, p. 72.

[g] Swann, T., place cited.

As in smelting high-silica ores some silicon will in any event go into the alloy; it appears that it will probably be best to make an alloy with 15 to 20 per cent silicon. In fact, utilization of ores high in silica appears to demand such a silicon content.

PRODUCTION OF SILICOMANGANESE.

Very little silicomanganese has been used in the United States, although it is widely used abroad, being made in large amounts in Sweden, France, and Switzerland. Harden [a] gives the Swedish production of ferromanganese as 2,000 tons per year and that of silicomanganese as 3,500 to 4,000 tons. He says the silicomanganese is made from a mixture of manganese ore and quartz. Lyon, Keeney, and Cullen [b] state that it is also made from rhodonite ore. Swedish practice is said [c] to utilize Bessemer and blast-furnace slags for the manufacture of silicomanganese.

While the demand for silicomanganese is still low in the United States, one firm, the Pacific Electro Metals Co., is making it. The ore used contains 40 per cent Mn, 23 per cent SiO, very little phosphorus, and practically no CaO, MgO, or Al_2O_3. Quartz is added to the ore to form a mixture of about 30 per cent Mn and 35 to 40 per cent SiO_2. A low-grade ore of this composition would be acceptable. Steel scrap is also added to the charge, and the alloy produced runs 50 to 55 per cent Mn. 20 to 25 per cent Si, 0.6 per cent C, 0.2 per cent P, and the remainder iron. The power consumption per long ton of alloy produced is said to be 5,800 kw.-hours, and the coke consumption 1,400 to 2,000 pounds in a 3,000-kw. furnace. The furnace is run without flux, low-ash coke being desirable, and the whole charge is reduced to metal, just as in ferrosilicon smelting. This would be possible only with a practically gangue-free ore.

The absence of slag avoids the usual loss of manganese in the slag incident to making ferro-alloy—the manganese recovery in making silicomanganese being 95 per cent. That is, with an ore of this grade, 20 per cent more manganese is recovered in alloy than when ferro-alloy is made—or 4 tons of ore made into silicomanganese gives as much usable manganese as 5 tons made into ferromanganese.

---

[a] Harden, J., Utilization of manganese ores in Sweden: Met. and Chem. Eng., vol. 17, 1917, p. 701.

[b] Lyon, D. A., Keeney, R. M., and Cullen, J. F., work cited.

[c] News item, Manganese from steel slags: Min. and Sci. Press, vol. 116, Apr. 6, 1918, p. 486.

As regards conservation, silicomanganese, rather than ferro, should therefore be made from ores low in lime and alumina.

The addition of iron is probably due to the buyer's specification. It is illogical to add iron, use power to melt it, and then have to ship it to the steel works where all it does is to chill the steel bath when the alloy is added. By leaving out the iron an alloy containing 68 per cent Mn, 2 per cent Fe, 29.5 per cent Si, 0.3 per cent C, and 0.23 per cent P could be produced and should be preferable to the alloy now made.

FUTURE OF HIGH-SILICA ORES DEPENDENT ON USE OF SILICOMANGANESE.

Any utilization of low-grade ores high in silica postulates that the steel industry can and will use silicomanganese.

Only a few brief calculations are required to prove that there is enough steel produced for castings, forgings, etc., to which both ferromanganese and ferrosilicon are added, to absorb a very large production of silicomanganese. Common sense and foreign practice, as well as actual large-scale experiments in this country, indicate not only that this alloy can be used, but that its use is distinctly desirable.[a]

It should be noted that the high-silicon alloys contain very little carbon, so that where recarburization is now done by ferromanganese or spiegel, other means of recarburization, such as the use of charcoal, coal, or coke, must be resorted to when silicomanganese is substituted for ferro or spiegel. The low carbon content would, on the other hand, often be a distinct advantage.

CARBON CONTENT OF MANGANESE ALLOYS.

The relation of the silicon to the carbon content in alloys containing 60 to 80 per cent manganese is shown by the curve in figure 13. Silicospiegels of lower manganese content will give a curve lying below this one, as iron takes up less carbon than manganese. The smelting temperature, as well as the practice with regard to carbon, affects the carbon content; but for alloys made by electric smelting at normal smelting temperature and with an excess of carbon, the curve is closely accurate. As regards alloys with less than 5 per cent silicon, the carbon content may vary about 0.75 per cent from the values shown by the curve. For alloys with 10 per cent silicon the deviation will be less than 0.5 per cent carbon; for those with 15 per cent silicon and higher, the deviation will be only 0.1 to 0.2 per cent carbon.

---

[a] See Swann, T., The development of the ferromanganese industry in the United States since 1914: Chem. and Met. Eng., vol. 19, 1918, p. 672.

REASONS FOR EXPERIMENTAL WORK.

It is natural that commercial electric-furnace plants should procure the highest grade raw material available. For this reason little is known about the actual working possibilities of low-grade manganiferous ores. If any experiments have been made on such ores, the records have not been published. Their behavior in the electric furnace can, to a certain extent, be predicted by a metallurgist skilled in smelting high-grade ores, but beyond a certain point, such predictions become little more than speculation.

As the United States has immense known deposits of low-grade ores, many of which are not amenable to ordinary methods of gravity concentration, the study of the smelting of such materials in the electric furnace, as herein described, was undertaken by the Bureau of Mines. It was felt that if information could thereby be secured that would make any portion of these immense ore reserves commercially available, a real contribution would be made to the country's resources.

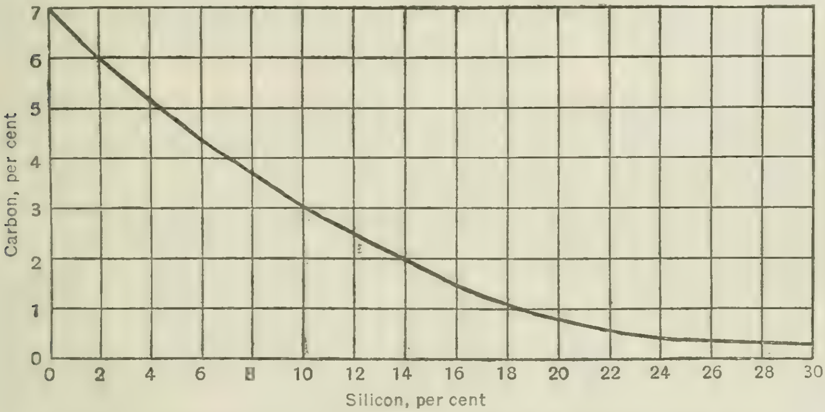

FIGURE 13.—Curve showing relation of carbon to silicon in ferro-silicomanganese alloys.

CLASSES OF LOW-GRADE ORES.

In the utilization of low-grade domestic ores, one would naturally start with the best of such ores—that is, those highest in manganese or lowest in silica, leaving the leaner ores until experience had been gained on the better ores. Attention should, then, be paid (1) to ores high in silica, low in manganese, and low in phosphorus, like the low-grade Butte ores; (2) to similar ores, but high in ratio of phosphorus to manganese, such as the Philipsburg ores; and (3) to those high in iron, such as the Cuyuna ores.

Exact analyses of the low-grade Butte ores are not available, but from Pardee's statements[a] we may assume that large tonnages of ore containing 25 per cent Mn, 40 per cent SiO, 3. 5 per cent Fe, approximately 5 per cent Al_2O_3, less than 0. 05 per cent P, 13 per cent loss on ignition, and only a trace of CaO, should be available. Obviously, such ore would be eminently fitted for the manufacture of silicomanganese, smelting without a slag, provided the alumina did not form a stiff slag of sufficient volume to give trouble. In attempting complete reduction of this ore some aluminium would probably be reduced into the alloy; but as this aluminium would act first in the deoxidizing work of the alloy in steel, being thus eliminated and at the same time protecting the manganese, its presence would probably not be detrimental. In fact silicomanganese-aluminum of high aluminium content is used abroad as a deoxidizer. The low phosphorus content of this ore and its low content of slag-forming constituents make it appear one of the most promising of the low-grade ore.

---

[a] Pardee, J. T., Manganese at Butte, Mont.: U. S. Geol. Survey Bull. 690, 1918, pp. 123–125.

The Philipsburg (Montana) ore is low in iron and the deposits are large. Special lots of low-grade Philipsburg ore have given the following analyses:

Results of analyses of low-grade Philipsburg (Montana), manganese ores.

Lot No.	Mn.	Fe.	SiO.	CaO-MgO-BaO.	Al_2O_3.	P.	Loss on ignition.
	Per cent.	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*	*Per cent.*
1	29.5	2.5	25.0	9.6	1.5	0.105	18.7
2[a]	26.6	2.8	35.9	6.7	5.8	.24	13.3
2b[b]	24.1	1.6	37.7	7.7	1.6	.37	12.7
2c[c]	28.7	2.4	28.5	4.1	3.4	.14	14.6
4[c]	26.4	2.5	36.4	7.5	1.9	.21	11.8

[a] Contained both lump and screenings. [b] Lump picked off grizzly. [c] Screenings, ½-inch screen.

The lump and screenings are obtained in the concentration (by hand picking and screening only) of the run-of-mine ore. The screenings are thought to constitute about 20 per cent of the ore as mined. This type of ore is too basic to allow running without a flux. The ratio of phosphorus to manganese content is high. There would then be two problems, namely, how to flux the ore and how to obtain an alloy low in phosphorus.

EXPERIMENTAL WORK.

To solve these and similar problems, advantage was taken of the electric-furnace equipment of the Bureau of Mines field office at Cornell University, Ithaca, N. Y. Small-scale experiments of this sort are admittedly inconclusive. They go a long way, however, toward replacing speculations and proving theories, and may safely be used as guides to large tests on a commercial scale.

The ores used in the tests were obtained from Philipsburg. Their analyses are given in the foregoing table.

DESCRIPTION OF FURNACE USED.

A single-phase furnace with conducting bottom, with a carbofrax (carborundum) lined shaft 14 inches in diameter and 18 inches deep, surmounted by a fire-brick shaft 12 inches deep, was built. The upper electrode was a square carbon 4 by 4 inches. The carborundum lining worked well, and required few repairs. What little patching was required was readily done with a cement of fine carborundum and water glass. Inasmuch as many of the slags were acid, a magnesite lining would not have been satisfactory. As the runs were intermittent a carbon lining would have been liable to oxidation while the furnace was cooling between runs. The furnace uses about 65 kw., or 1,450 amperes at 50 volts, with an average power factor of 90.

PRELIMINARY TESTS.

Some continuous runs of about 14 hours were made with screenings that passed a ½-inch screen, keeping the shaft full of ore and tapping at intervals. The ore contained much fine dust, and was too fine to bridge well; hence in most runs the furnace was run discontinuously, one charge being smelted and tapped before a succeeding charge was added. The tap hole was closed with a charcoal plug and a wad of fire clay rammed over the plug. In tapping, the fire clay was dug out, when the charcoal could be readily barred out. Starting with a cold furnace, it took about 8 hours to smelt three batches of ore each of 100 pounds. To obtain a slag low in manganese, and a high recovery of manganese in the alloy, it was necessary to run the furnace hot, using not less than 60 to 65 kw. A fairly fluid slag and a considerable excess of reducer were also required. The excess of reducer does not usually come out of the furnace with the slag, but accumulates from heat to heat in the furnace bottom, so that in commercial operation after the first few heats considerably less reducer would be needed than was required in the experimental work. The excess of coke or charcoal holds some slag and metal in its pores so that the "dross" scraped out of the furnace after a series of runs is always higher in manganese than the slag. In commercial operation the "dross" would remain in the furnace and the metal lost in operation on an experimental scale would be recovered.

The manganese content of the slag on the first heat in a cold furnace is always higher than on subsequent heats, even though the slags may be tapped at equal temperatures.

STANDARDIZATION OF FURNACE.

For these reasons the manganese recovery to be expected in the commercial operation would be higher than that obtained in small-scale experiments. Power consumption and electrode consumption in a small furnace, when the furnace is started cold and only a short run is made will be much higher than in commercial practice. A rough factor for calculating power consumption in large-scale operations can be obtained by running a high-grade ore in the small furnace and comparing the power used with that required in commercial operation on such an ore.

Such a standardization was made with a high-grade ore—Philipsburg concentrates—analyzing 47.7 per cent Mn, 1.9 per cent Fe, 8.7 per cent SiO_2, 2.1 per cent CaO, 1.1 per cent Al_2O_3, 0.08 per cent P, 14.2 per cent loss on ignition. The usual basic slag was made. The results follow:

Run 52.—Three charges, each consisting of 100 pounds of ore, 3 pounds of steel scrap, 14 pounds of coke (metallurgical coke, $\frac{1}{4}$-inch mesh; analyzing 4.75 per cent H_2O, 10.3 per cent ash, of which 1.4 per cent was Fe, 4.3 per cent SiO_2, 2.5 per cent Al_2O_3, 0.8 per cent CaO), 14 pounds of charcoal (crushed to about $\frac{1}{2}$ inch), and 20 pounds of $CaCO_3$ (marble, crushed to $\frac{1}{2}$ inch). Time of run, 8 hours 10 minutes; power used, 440 kw.-hours (180 kw.-hours, first heat; 140, second heat; 120, third heat). Obtained 158.2 pounds of metal, containing 82.0 per cent Mn, 2.0 per cent Si, 0.12 per cent P; 84.3 pounds of slag, containing 8.5 per cent Mn; and 11 pounds of dross containing 34.5 per cent Mn. Mn distribution: In metal 90.5 per cent, in slag 4.2 per cent, in dross 2.7 per cent, volatilization and dusting loss 2.6 per cent. Recovery of P in metal, 79 per cent.

As, in commercial practice, not over 4,000 kw.-hours per long ton of alloy would be required on such an ore in a 3,000-kw. furnace, or 2.2 kw.-hours per pound of metallic manganese in the alloy, and as the last two taps in the small furnace used 3.0 kw.-hours per pound of manganese in the alloy, it appears that 75 per cent of the power used per pound of alloy in the experimental furnace on the last two taps of a run on 300 pounds should give the approximate power required commercially. A previous run (No. 50) on this high-grade ore, using the same flux, and adding 5 pounds of mill scale, to supply iron, 10 pounds of burnt lime, and 33 pounds of charcoal per 100 pounds of ore, gave an alloy of 77 per cent Mn, 2 per cent Si, 0.16 per cent P, with a manganese recovery of 79 per cent. Some of the heats were not given enough power, and some of the slags were high in manganese. It was calculated that under correct operating conditions 88 per cent of the manganese would have been recovered. This calculation was checked by the results of run 52. The results of these runs indicated that a mixture of half coke and half charcoal was as good a reducer as all charcoal.

FLUXING TESTS WITH HIGH-SILICA ORE.

A series of fluxing tests was made with a high-silica ore, namely, the Philipsburg screenings, lot 4 (see table of analyses), containing 26.4 per cent Mn, 2.5 per cent Fe, 36.4 per cent SiO_2, 7.5 per cent CaO+MgO, 1.9 per cent Al_2O_3, and 0.21 per cent P, with 11.8 per cent loss on ignition. In all the runs 10 pounds of coke and 10 pounds of charcoal were used per 100 pounds of ore, except in run 73 in which 15 pounds of each were used. No steel scrap was added except 0.5 pound per 100 pounds of ore in run 70. The results of the runs are summarized as follows:

Results of fluxing tests with high-silica ore.

CHARGES AND POWER CONSUMPTION.

Run No.	CaO per 100 pounds of ore.	CaF_2 per 100 pounds of ore.	CaO per pound of silica, including CaO equivalent to bases in ore.	CaF_2 per pound of silica.	Ore used.	Time of run.	Power consumed.
	Pounds.	*Pounds.*	*Pounds.*	*Pounds.*	*Pounds.*	*H. min.*	*k.w.-hours.*
70	72	....	1.30	....	300	8 20	522
69	28	....	.60	....	300	7 20	450
67	19	7	.36	0.19	300	7 25	453
68	....	14.5	.20	0.40	300	7 20	452
73	None till end, then 3.5.	10	None till end, then 0.30.	0.28	200	7 8	469

PRODUCTS AND POWER CONSUMPTION.

Run No.	Alloy made per 100 pounds.	Analysis of alloys.			Slag made per 100 pounds.	Mn in slag.	P in slag.	Kw.-hours per pound of alloy, including all heats.	Kw.-hours per pound of Mn, including all heats.
		Mn.	Si.	P.					
	Pounds.	*Per cent.*	*Per cent.*	*Per cent.*	*Pounds.*	*Per cent.*	*Per cent.*		
70	23.8	71.0	13.0	0.60	93	6.0	0.04	7.3	10.2
69	18.4	71.4	11.6	.73	61	11.5	.06	8.2	11.4
67	21.9	73.0	13.0	.68	65	14.0	.04	6.9	9.5
68	26.3	71.0	15.5	.60	59	12.0	.04	5.75	8.1
73	26.7	72.7	19.8	.57	17	23.1	.04	8.8	12.0

DISTRIBUTION OF ELEMENTS IN PERCENTAGES OF TOTAL.

Run No.	In metal.			In slag.		In dross.		Loss.	
	Mn.	Si.	P.	Mn.	P.	Mn.	P.	Mn.	P.
70	63	8½	68	21	18	....	....	16	13
69	50	6	64	26	18	7	5½	17	12½
67	60	8	70	34	3½	....	(a)	3	13½
68	71	11	75	27	11½	....	(a)	2	13½
73	73	14½	72½	21	5	6	10½	0	12

[a] Amount of dross small; included in weight of slag and in sample of slag for analysis.

More power was used in run 70 than in the others because of the large amounts of $CaCO_3$ charged. The slag was very stiff in run 69 and an arc had to be used to tap. More power might have given a slightly better recovery.

In run 73 more reducer was used than in the other runs, and no flux was added till the end. Two batches of 100 pounds of ore each were added before tapping. The metal tapped, but the slag was too stiff to run; the fluxes were than added, and 45 kw.-hours were used (included in the 469 total in the table) to get the slag out. It then tapped well.

With the same power input for runs 69 and 67 as for run 70, run 67 would have given better results than run 70, but the results of run 69 would not have been as satisfactory as for run 70, the slag being too stiff. The acid slags containing fluorspar, however, tapped well.

Considering the cost of power and fluxes, it appears that the acid slags of runs 67 and 68 were preferable to the basic slag of run 70. It is noteworthy that the basic slag did not prevent the reduction of silicon in run 70. The recovery of manganese was higher in run 73, operating without flux until the furnace got clogged with slag and then fluxing out the slag, but the power consumption was too high. The flux of run 68—all CaF_2, except for the CaO in the ore itself—gave the best results metallurgically. The slag of run 68 was very fluid, and some of the CaF_2 could probably have been replaced by CaO, using a flux intermediate between those of runs 68 and 73. Slags made with fluorspar, are, for an equal Ca content, much more fluid than those made with lime.

ADVANTAGES OF ACID SLAG.

The percentage of manganese is somewhat lower in a strongly basic slag than in an acid slag, but the weight of slag formed is much greater. Hence, the weight of manganese lost in the slag is greater with the basic slag, when the SiO_2 content of the ore is much above 25 per cent. When the ore is fluxed to a basic slag the cost of handling and disposing of the excess slag formed must be considered. The cost of the lime flux for the basic slag would, of course, be lower than that of the fluorspar required to produce an acid slag fluid enough to tap; although in a large furnace, where the tap hole would not freeze as readily as in the small one when tapping a sticky slag, the amount of fluorspar could doubtless be materially reduced. The weight of manganese lost in the slag is greater with the basic slag, when the SiO_2 content of the ore is much above 25 per cent. When the ore is fluxed to a basic slag the cost of handling and disposing of the excess slag formed must be considered. The cost of the lime flux for the basic slag would, of course, be lower than that of the fluorspar required to produce an acid slag fluid enough to tap; although in a large furnace, where the tap hole would not freeze as readily as in the small one when tapping a sticky slag, the amount of fluorspar could doubtless be materially reduced.

It will be noted that in the best run (No. 68), the ratio of phosphorus to manganese in the alloy was 1:118, whereas in the ore the ratio was 1:125. Thus it appears that phosphorus and manganese were recovered in about the same proportion.

TEST OF REDUCERS.

On account of the variation in commercial practice as to the reducer used, anthracite, coke, or a mixture of coke and charcoal being used by different firms, a series of tests was made with reducers.

Some preliminary runs in a small furnace taking only about 20 pounds of ore per charge were made. The Philipsburg screenings used contained 29.5 per cent Mn, 2.5 per cent Fe, 25.0 per cent SiO_2, 9.5 per cent $CaO + MgO$, 1.5 per cent Al_2O_3, and 0.105 per cent P, and showed 18.7 per cent loss on ignition. The following results were obtained, the same power input being used in each run:

Results of tests of reducers in small electric furnace.

	Run 20.	Run 21.	Run 22.
Charge:			
Ore .... pounds..	20	20	20
Coke .... do....	6	.....	.....
Anthracite .... do....	.....	5	.....
Charcoal .... do....	.....	.....	5
Calcium fluoride (CaF_2) .... do....	2½	2½	2½
Analysis of alloy made:			
Manganese (Mn) .... per cent..	74.0	77.3	75.3
Silicon (Si) .... do....	13.0	4.6	12.5
Phosphorus (P) .... do....	0.24	0.32	0.20
Recovery in alloy:			
Manganese .... do....	66	43	66
Phosphorus .... do....	57	52	40

Anthracite gave the lowest recovery. The recovery with coke was the same as that with charcoal.

Runs 48 and 49 were then made in the larger furnace. The slag used was calculated for a ratio of CaO to SiO_2 of 0.4:1, which in the later fluxing tests was shown to be too stiff a slag for good recovery. Seventy-five pounds of ore was used per charge and the furnace was run 12 to 13 hours. The data on the average ore mixture used, the composition of the charge, and the results follow:

Data on reducer tests with large electric furnace.

	Run 48.	Run 49.
Ore mixture:		
Mn .... per cent..	28.7	29.0
Fe .... do....	2.4	2.2
SiO_2 .... do....	28.5	26.5
CaO .... do....	4.1	4.2
Al_2O_3 .... do....	3.3	2.7
P .... do....	.14	.16
Loss on ignition .... do....	14.5	16.5

Charge (weight of constituent to each 100 pounds of ore):		Run 48.	Run 49.
$CaCo_3$	pounds	12	12
Charcoal	do	12.3	None.
Coke	do	None.	13.5
Quantity of ore charged	do	900	825
Time of run	hours	12.25	13
Power used	kw.-hours	714	710
Weight of alloy made	pounds	182.0	146.1
Composition of alloy:			
Mn	per cent	74.0	76.0
Si	do	10.0	8.0
P	do	.34	.38
Weight of slag made	pounds	496	462
Manganese in slag	per cent	16.4	18.3
Recovery in alloy:			
Mn	do	52	46
P	do	49	42

As regards the two tests, although the ore used in run 49 was slightly higher in manganese and lower in silica, and although more power was used per unit weight of ore, charcoal gave better results than coke. However, neither result was satisfactory, the slag being too acid and holding up too much manganese. A slag of the same acidity made with fluorspar would have shown better results. The amount of reducer in both tests was too small for good results.

COMMENTS ON USE OF RHODOCHROSITE.

As the lot of ore used in run 48 was exhausted before run 49 was finished, the last few charges in run 49 were made with a mixture of uncalcined rhodochrosite and Philipsburg lump (lot 2b). On account of the high loss (31 per cent) on ignition of the rhodochrosite, it gave trouble, even when mixed with the other ore. The large volume of CO_2 given off by the raw rhodochrosite tended toward excessive electrode consumption, and probably also toward wasting of the reducer in the charge. If ore so high in volatile matter falls into the melt before it has been thoroughly calcined in the shaft, the furnace will "blow," or form a "volcano."

For these reasons, and because it should be cheaper to calcine by fuel heat than to expel CO_2 by electric heat in the furnace, it will probably be found desirable to calcine rhodochrosite ore before charging it into the electric furnace. The ore decrepitates somewhat on calcination, and the product may have to be screened, the coarse material being then smelted continuously and the fines fed to a separate furnace as fast as the melt will take the ore, but without keeping the shaft piled full, in order to keep down dust losses. Rhodochrosite, may, in fact, be too high grade an ore for electric smelting without being mixed with ores containing more slag-forming constituents.

The analysis of the rhodochosite used, after complete calcination, was 53.5 per cent Mn, 2.9 per cent Fe, 8.8 per cent SiO_2, 1.65 per cent

$CaO+MgO$, 1.4 per cent Al_2O_3, and 0.04 per cent P. Fluxed to the ordinary basic slag and with suitable addition of steel scrap, this would give only about 1,050 pounds of slag per long ton of 80 per cent ferromanganese produced. Compared to the 40 per cent Mn 20 per cent SiO_2 ore in common use, which, when similarly slagged, gives about 4,000 pounds of slag per long ton of alloy, the rhodochrosite may cause too short a column of slag, and hence, at the low voltage of a ferromanganese furnace, may not present sufficient electrical resistance to develop enough heat.

In the ordinary three-phase, nonconducting-bottom furnace, the current may flow from one electrode to the other two by two paths—(1) through slag entirely, or (2) from the electrode straight down through the slag to the alloy beneath and then through the alloy to directly beneath the other electrodes, and up through the slag again to the other electrodes. With too short a column of slag, path 1 presents the least resistance and will result in the generation of heat in the slag in three spots only. With a deeper slag column, the resistance of the path from one electrode to another through the slag only is less than that from electrode to electrode by way of slag, alloy, and slag. If the power input is too low, the large mass of metal below the slag may get cool enough to make tapping difficult, and the slag itself may not get hot enough for complete reduction of manganese, resulting in high manganese loss in the slag. If, in order to develop enough heat, the furnace is run as an arc furnace instead of as a resistance furnace, local overheating will occur and manganese will be lost by volatilization. In this connection, the comments of Bardwell[a] on smelting rhodochrosite are of interest.

TESTS OF EFFECT OF SIZE OF REDUCER CONSTITUENTS.

Runs 50 and 52, previously mentioned, had shown that the mixture of half coke with half charcoal worked as well as all charcoal. When all one-fourth-inch coke was used as reducer, some of the excess of fine coke tended to emulsify with the slag, coming out with it and making it harder to tap. When half coke and half charcoal was used this was not the case. It was thought that larger coke might give better results, so a series of runs was made to test this assumption and to compare the results from coke and from anthracite with the results of runs in which a mixture of half coke with half charcoal was used. The results of the tests are presented in the table following. The ore used was Philipsburg screenings, lot 4, containing 26.4 per cent Mn, 2.5 per cent Fe, 36.4 per cent SiO_2, 7.5 per cent $CaO+MgO$, 1.9 per cent Al_2O_3, and 0.21 per cent P, with a 11.9 per cent loss on ignition. It was crushed to pass a one-half-inch screen. As a flux, only 14.5 pounds of CaF was used per 100 pounds of ore. Three heats were made on each run; that is, 300 pounds of ore was used in each run.

---

[a] Bardwell, E. S., Discussion; Bull. 43, Am. Inst. Mining Eng., Nov., 1918, p. 1651.

Results of tests of reducers to determine effect of size of constituents.

Run No.	Coke[a] ($\frac{1}{4}$-inch).	Charcoal ($\frac{1}{2}$-inch).	Coke[b] (1 to $\frac{1}{4}$ inch).	Coke[b] (less than $\frac{1}{4}$-inch).	Anthracite[c] ($\frac{3}{4}$ to $\frac{1}{4}$ inch).	Anthracite[d] (less than $\frac{1}{4}$-inch).	Recovery of Mn.	Recovery of P.
	Pounds.	*Pounds.*	*Pounds.*	*Pounds.*	*Pounds.*	*Pounds.*	*Per cent.*	*Per cent.*
68	10	10					71	75
71			21				54	68
74				21			35	
72					21		46	65
75						21	32	53

Run No.	Time.	Power used, kw.-hours.	Alloy made.	Composition of alloy.			Slag made.	Mn in slag.	P.	Dross.		
				Mn.	Si.	P.				Weight.	Mn content.	P content.
	Hr. min.		*Lbs.*	*P. ct.*	*P. ct.*	*P. ct.*	*Lbs.*	*P. ct.*	*P. ct.*	*Lbs.*	*P. ct.*	*P. ct.*
68	7 20	452	78.8	71.0	15.5	0.60	177	12.0	0.04	5	(e)	(e)
71	7 15	450	60.8	71.0	16.5	.71	156	15.4	.06	21	14.9	0.13
74	6 45	450	57.6	73.5	13.2	(f)	183	15.0	(f)	29	21.0	(f)
72	7 0	450	50.7	72.0	15.0	.81	200	15.6	.03	29	13.9	.31
75	6 55	450	[g] 33.6	73.6	11.5	1.00	205	13.1	.03	66	20.7	.14

[a] Metallurgical coke, 4.75 per cent H_2O, 10.3 per cent ash.
[b] Gas coke, 13.8 per cent ash.
[c] 4.5 per cent H_2O, 8.5 per cent ash.
[d] 3.5 per cent H_2O, 13 per cent ash.
[e] Included in slag.
[f] Not determined.
[g] The fine coal lying on the slag at the end of a tap ran out with the slag, leaving less excess reducer in the furnace on the last two heats than in the other runs. In run 75, 20 to 24 pounds of dross was obtained on each tap. In the other runs the dross was all taken out after the last tap only.

These results indicate that anthracite is not as good a reducer as coke, perhaps because anthracite is less porous, giving less area of contact with the melt. It would be interesting to know whether the difference in the performance of coke and anthracite would be as marked in a large furnace as it was in the experimental one. The experimental work indicates that, at least with a highly siliceous ore, the mixture of coke and charcoal used by the Noble Electric Steel Co. should be highly desirable where charcoal is available.

SLAG-SMELTING TESTS.

In the summer of 1917 Prof. F. F. McIntosh, consulting chemist of the Bureau of Mines, and assistants, made some tests at the Carnegie Institute of Technology, Pittsburgh, Pa., on making manganese alloys from slags. A 50-kw. furnace was used. The slags were not fluxed, and as the iron and manganese were reduced out the slags became stiff, so that the furnace had to be torn down after each tap. The results, therefore, could hardly be applied directly to commercial practice. Table 30 gives a summary of some of the results.

TABLE 30.—*Results of slag smelting tests in electric furnace.*

Run No.	Slag or slag mixture.	Kind of slag.	Analysis of slag or mixture.					Reducer.		Time.	Kw.-hours used.	Alloy made.	Composition of alloy made.				Recovery of Mn.
			Mn.	Fe.	SiO_2.	CaO.	Al_2O_3.	Coke.	Charcoal.				Mn.	Fe.	Si.	P.	
	Lbs.		*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*Lbs.*	*Lbs.*	*Hr. min.*		*Lbs.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
G	240	Bessemer ladle	8.5	13.5	57	2.5	7	44	....	7 45	260	59.5	23	50	26	....	65
H	1,090	Bessemer ladle plus basic open-hearth.	11	21	42	12	10	178	....	22 0	900	108.5	14.5	71	12	0.50	13
J	536	Bessemer ladle plus manganese steel plus spiegel cupola slags.	24	7	42	6	11	20	116	14 45	650	126.5	52.5	26	19	.11	53
K	548	....do	22	5.5	45	6	12	....	126	13 15	660	142.0	53	28	21	.14	63
L	280	Manganese steel plus spiegel cupola slag.	26	6.5	40	[a] 3	12	122	....	6 15	320	51	51.5	27	17	.06	36

[a] 19 pounds of CaO also added to the charge.

Runs K and L indicated that charcoal was a better reducer than coke. Evidently such manganese-bearing slags need to be fluxed, so as to make a fluid slag, as they contain 11 to 22 per cent $Al_2O_3 + CaO$, but those of about 25 per cent Mn content could be used to make silicomanganese. The basic open-hearth slag used in run H was rather high in phosphorus. The other slags were low in phosphorus, and hence made alloys very low in that element.

EFFECT OF PHOSPHORUS CONTENT OF ORES.

In the experimental runs low-grade ores high in phosphorus gave an alloy rather high in that constituent. If smelted with the shaft kept full of ore, the alloy might be still higher in phosphorus. A considerable loss of phosphorus by volatilization took place, and the slags were always rather low in phosphorus, no matter what the original content in the ore was, or whether the slag was acid or basic. The percentage recoveries of phosphorus and of manganese in the alloy were usually within a few per cent of each other.

Lyon, Keeney, and Cullen [a] state that in the electric smelting of ferromanganese all the phosphorus in the charge goes into the metal, and that in making ferrosilicon most of it does, owing to the strongly reducing conditions.

Newton [b] makes a similar statement in regard to blast-furnace ferromanganese smelting. Continuous smelting with the shaft kept full of ore will tend to condense any volatilized phosphorus, so that the losses found in the experimental smelting by batches may not occur in normal practice. Doubtless the lower the ore is in phosphorus the less will escape. The remarks cited are, of course, based on normal practice with low phosphorus ores. However, W. W. Clark, former manager of the Noble Electric Steel Co., states [c] that phosphorus is volatilized in the Noble ferromanganese furnace.

Lonergan [d] cites a test run at the Iron Mountain Alloy Co. works on ore having a very low phosphorus content (less than 0.04 per cent P) in which 52.6 per cent of the phosphorus contained in the ore, coal, and limestone was recovered in the ferromanganese, 14.75 per cent in the slag, and 6.06 per cent in "dirty metal and slag," leaving 31.4 per cent phosphorus lost by volatilization and in dust. Lonergan concluded that the greater part of this loss is purely mechanical, the phosphorus in the dust being retained in its original form. He assumes that any volatilized phosphorus would go off as phosphine.

The writers see nothing to prevent elemental phosphorus being volatilized, as it is in electric-furnace production of phosphorus, and burning to the oxide as soon as it reaches the air.

---

[a] Lyon, D. A.; Keeney, R. M.; and Cullen, J. F.: The electric furnace in metallurgical work: Bull. 77, Bureau of Mines, 1917, pp. 144, 166.

[b] Newton, E., Manganiferous iron ores of the Arizona District, Minn.: Bull. 5, Minn. School of Mines Exp. Station, Univ. of Minn., 1918, p. 75.

[c] Personal communication.

[d] Lonergan, J., Eliminating phosphorus and sulphur in electric ferromanganese furnaces: Met. and Chem. Eng., vol. 20, 1919, p. 245.

If 31.4 per cent of the manganese also was lost in dust, to say nothing of volatilization, then Lonergan's assumption that phosphorus was lost only mechanically would be justified; but according to Keeney,[a] the average loss by volatilization and dusting in the Iron Mountain Alloy Co.'s practice is 7.3 per cent. It would therefore appear that even on ores so low in phosphorus an appreciable amount of the phosphorus is volatilized.

TESTS WITH HIGH-PHOSPHORUS ORES.

Runs 60 and 61 were made with Philipsburg lump ore, lot 3b, containing 24.1 per cent Mn, 1.6 per cent Fe, 37.7 per cent SiO_2, 7.7 per cent CaO + MgO, 1.6 per cent Al_2O_3, and 0.37 per cent P, with 12.8 per cent loss on ignition. The procedure and results were as follows:

Results of run 60—discontinuous smelting, each charge being tapped before adding the next.

[Furnace cold at start.]

	Charge a.	Charge b.	Total for run.
Make-up of charge:			
Ore .... pounds..	100	100	....
$CaCO_3$ (0.32 pound CaO+O per pound of SiO_2) .... do....	8	8	....
CaF_2 (18 pounds per pound of SiO_2) .... do....	7	7	....
Coke .... do....	7.5	8.5	....
Charcoal .... do....	7.5	8.5	....
Steel scrap .... do....	0.5	None.	....
Time of run	3 h. 25 m.	2 h. 25 m.	5 h. 50 m.
Power used .... kw.-hours..	149	130	279
Metal tapped .... pounds..	6.0	18.7	24.7
Slag tapped[a] .... do....	65.0	70.5	135.5

	Per cent.		Per cent.
Analysis of metal:		Distribution of metal:	
Mn	66.0	Mn	34.0
Si	13.5	P	27.5
P	0.71		
Analysis of slag:		Distribution of slag:	
Mn	15.0	Mn	42.5
P	0.7	P	14.5

[a] Some slag stuck in furnace.

Results of run 61—continuous smelting, shaft being kept full.

[Total charge in pounds: Ore, 400: $CaCO_3$ (0.35 pound CaO per pound of SiO_2), 40; CaF_2 (1.18 pound per pound of SiO_2) 28; coke, 38.5; charcoal, 38.5.]

Tap No.	After.	Kw.-hours used.	Metal tapped.	Analysis of metal.			Slag tapped.	Analysis of slag.	
				Mn.	Si.	P.		Mn.	P.
	Hr. min.		*Pounds.*	*Per ct.*	*Per ct.*	*Per ct.*	*Pounds.*	*Per ct.*	*Per ct.*
1	2 30	142	13.5	66.5	11.6	0.80	92.8	17.1	0.07
2	1 50	141	30.1	70.5	16.5	.95	84.0	13.2	.09
3	2 55	141	31.1	70.0	18.3	.93	30.0	11.8	.12
4	40	28	.8	....	....	....	[a]42.5	10.5	.15
Total	7 55	452	75.8	....	....	....			
Average	....	....	....	69.0	16.0	.93	249.3	14.0	.095

Distribution.	Mn.	P.
	Per cent.	*Per cent.*
Metal	54.5	57.5
Slag	35.5	21.5
Loss	10.5	21

[a] Includes dross.

---

[a] Keeney, R. M., The manufacture of ferroalloys: Bull. Am. Inst. Min. Eng., August, 1918, p. 1333.

These results, compared with those from run 68, show that the power supply was insufficient for good recovery of the manganese, 452 kw.-hours being used on 400 pounds of ore in run 61 and 452 on 300 pounds of ore in run 68. The reducer was probably not in too small excess, and a slag higher in CaF_2 would have given better results. Under proper conditions this ore should give the 71 per cent recovery of run 68, when the increased reduction of manganese and silicon should bring the phosphorus down to about 0.75 per cent.

TESTS ON VOLATILIZATION OF PHOSPHORUS.

As the volatilization of phosphorus is probably due to to its reduction from the calcium phosphate in the slag to elemental phosphorus, which will then either volatilize or be absorbed by the metal, it would appear that if only enough reducer were used to reduce Fe and Mn to FeO and MnO, and P_2O_5 to P, and the charge were held molten for a time before more reducer were added to cause a metal fall, the elimination of phosphorus would be favored. With this consideration in mind, run 62 was made.

In this run each of the three batches charged consisted of 100 pounds of lot 2b ore, 10 pounds of $CaCO_3$, 7 pounds of CaF_2, and 2 pounds each of coke and charcoal. After the batch was melted it was kept molten for 20 minutes, then 8 pounds each of charcoal and coke were added. The results were as follows:

Results of run 62.

Tap.	Time.		Kw.-hours used.	Quantity of metal obtained.	Analysis of metal.			Quantity of slag obtained.	Analysis of slag.		Quantity of dross obtained.	Analysis of dross.	
					Mn.	Si.	P.		Mn.	P.		Mn.	P.
	Hr.	*min.*		*Lbs.*	*P. ct.*	*P. ct.*	*P. ct.*	*Lbs.*	*P. ct.*	*P. ct.*	*Lbs.*	*P. ct.*	*P. ct.*
a	3	5	182	13.5			1.29	67	13.3	0.06			
b	2	35	131	19.8			.97	78	11.8	.06			
c	2	20	130	15.0			1.16	70	11.7	.10	27	17.1	0.16
Total	8	0	443	48.3				215					
Average					71.0	14.0	1.12		12.6	.07			

Distribution:

		Mn.	P.
Metal	per cent	42	49
Slag and dross	do	43	31
Loss	do	15	20

As there was a considerable excess of reducer left after each tap, which would cause too early a metal fall and collect phosphorus, another run (No. 63) was made, omitting the reducer from the charge till all had been melted 20 minutes (except on the first charge), letting the excess reducer remain in the furnace after each tap, and then adding 10 pounds of coke and 10 pounds of charcoal. Five pounds of NaCl was added to each charge. Otherwise the charges were as in run 62. The results obtained were as follows:

Results of run 63.

Tap.	Time.	Kw.-hours used.	Quantity of metal obtained.	Analysis of metal.			Quantity of slag obtained.	Analysis of slag.	
				Mn.	Si.	P.		Mn.	P.
	Hr. min.	*P. ct.*	*Lbs.*	*P. ct.*	*P. ct.*	*P. ct.*	*Lbs.*	*P. ct.*	*P. ct.*
a	3 15	182	13.0	......	......	1.42	72	12.0	0.02
b	3 30	136	20.6	......	......	.75	81	11.9	.03
c	2 15	131	21.2	......	......	.80	93	11.0	*a*.03
Total	9 0	449	54.8	......	......	......	246	......	......
Average	......	......	......	71.6	14.5	.93	......	11.5	.03

a Slag included in dross.

Distribution:

		Mn.	P.
Metal	per cent	54.5	46
Slag	do	39.5	6
Loss	do	6	48

On the assumption that a more acid slag might help, another run (No. 64) was made, the total charge being practically the same as in runs 62 and 63, but divided as follows:

Charge for run 64, in pounds.

	Batch **a**.	Batch **b**.
Added at start:		
Ore	100	100
Charcoal	4	None.
$CaCO_3$	3	5
CaF_2	2	1
Added after heating 20 minutes:		
Charcoal	6	10
Coke	10	10
$CaCO_3$	7	8.5
CaF_2	5	6

Results of run 64.

Tap.	Time.	Kw.-hours used.	Quantity of metal obtained.	Analysis of metal.			Quantity of slag obtained.	Analysis of slag.		Dross.
				Mn.	Si.	P.		Mn.	P.	
	Hr. min.		*Pounds.*	*Per ct.*	*Per ct.*	*Per ct.*	*Pounds.*	*Per ct.*	*Per ct.*	*Lbs.*
a	4 30	210	9.3	68.0	12.8	1.45	65	18.0	0.11	......
b	2 35	148	10.9	71.8	11.8	1.57	63	18.0	.09	17
Total	7 5	358	20.2	......	......	......	128	......	......	......
Average	......	......	......	70.0	12.5	1.52	......	18.0	.10	......

Distribution:

		Mn.	P.
Metal	per cent	29.5	42
Slag	do	48	17.5
Loss	do	22	40.5

As the results of this run were not promising, another run (No. 59) was made in which the excess reducer was carefully scraped out after each tap, so that only the four pounds of reducer charged with the ore, as in run 62, would be present.

In run 59, the charge at the start consisted of 100 pounds of ore (lot 2b), 8 pounds of $CaCO_3$, 5 pounds of CaF_2, and 4 pounds of charcoal. After all was melted for 20 minutes, 6 pounds of charcoal and 6 pounds of coke were added; then the furnace was tapped. This procedure was repeated four times. The results follow:

Results of run 59.

Tap.	Time.	Kw.-hours used.	Quantity of metal recovered.	Analysis of metal.			Quantity of slag and dross recovered.	Analysis of slag and dross.	
				Mn.	Si.	P.		Mn.	P.
	Hr. min.		*Pounds.*	*Per ct.*	*Per ct.*	*Per ct.*	*Pounds.*	*Per ct.*	*Per ct.*
a	4 0	200	12.5	66.0	11.9	0.70	83.3	17.0	0.10
b	2 50	130	10.2				80.5		
c	2 45	116	9.6	66.2	12.3	.64	85.7	17.1	.06
Total	9 35	446					249.50		
Average			32.3	66.0	12.0	.68		17.0	.09

Distribution:

		Mn.	P.
Metal	per cent	29.5	19.5
Slag	do	58.5	20
Loss	do	12	60.5

These results, owing to the poor slag and the lack of sufficient excess reducer, are far from satisfactory as regards recovery of manganese, but they are somewhat promising as regards volatilization of phosphorus.

The low recoveries of manganese in this series were doubtless largely due to the ore not being properly fluxed. This defect was not realized till the later runs on fluxing were made. However, it is evident that one requirement for high recovery of manganese, namely, an excess of reducer, and one for high volatilization of phosphorus, namely, no excess reducer beyond that necessary to form MnO, FeO and P (that is, no metal fall to collect phosphorus), are incompatible. It is hardly practicable to clean the furnace from all excess carbon after each tap.

DEPHOSPHORIZATION AND SMELTING IN TWO STAGES.

Therefore, the only procedure metallurgically possible seems to be to carry out the dephosphorization in one furnace, running with just enough reducer to form MnO, FeO, and P, and holding the charge molten for a while to allow the phosphorus to escape; then to tap the hot slag into a second furnace, add excess reducer, and smelt the previously dephosphorized slag.

As it would be difficult to determine the exact point where there is complete reduction to MnO, FeO, and P, and as the electrode will also supply a little carbon, it would probably be more feasible to provide the dephosphorizing furnace with metal and slag taps, the latter being at a higher level. The slag can thereby be tapped into the second furnace, and a very little high-phosphorus metal can be taken either at each slag tapping or at longer intervals. This scheme would eliminate phosphorus not only by volatilization but also, in part, by concentration into the first metal to fall.

That this method is metallurgically possible is shown by the results of runs 56 and 57, presented in the following tabulations:

Charges used in run 56.

[Figures in pounds.]

Charge.	Ore.[a]	Charcoal.	Coke.	$CaCO_3$.	CaF_2.	Steel scrap.	Mill scale.
a	100	9.5	9.5	15	6	....	....
b	100	2.5	2.5	8	5	10	....
c	100	2.5	2.5	8	5	....	....
d	100	5.5	5.5	6	5	....	15
e	75	4	4	5	4	....	11

[a] From lot 2b.

NOTES.—
a. Run to reduce considerable Mn.
b. Run to reduce very little Mn, steel scrap added to collect P.
c. Run to reduce very little Mn.
d. Run to reduce very little Mn, and leave Fe in slag from mill scale.
e. Run to reduce very little Mn, but all the mill scale.

Charges used in run 57.

Charge.	Ore.	Charcoal.	$CaCO_3$.	NaCl.
a	50	1.9	6.6	4.3
b	50	1.9	6.6	4.3

NOTES.—
a. Run to reduce very little Mn.
b. Run to reduce a good deal of Mn.
It was hoped that the addition of NaCl would form phosphorus chlorides or oxychlorides that might be more readily volatile.

Results of runs 56 and 57.

Run.	Time.		Kw.-hours used.	Quantity of alloy made.	Analysis of alloy.		Slag.	Analysis of slag.	
					Mn.	P.		Mn.	P.
	Hr.	*min.*		*Pounds.*	*Per ct.*	*Per ct.*	*Pounds.*	*Per ct.*	*Per ct.*
56a	3	15	136	13.2	66.5	0.96	74.8	15.8	0.05
56b	1	25	73	12.8	41.1	1.02	99.5	22.0	.12
56c	1	40	60	2.0	69.6	1.41	87.0	23.0	.11
56d	1	20	59	2.6	18.8	1.23	86.0	21.6	.10
56e	2	15[a]	105	11.1	41.8	1.32	59.0	21.6	.05
57a	1	10[b]	41	.4	32.3	1.07	49.0	24.0	.09
57b	2	35	88	9.0	50.1	.39	41.0	16.6	.11

[a] Furnace cold at start.

[b] Furnace hot at start.

Results of runs 56 and 57—Continued.

DISTRIBUTION OF MN. AND P.

Run.	Mn.			P.		
	Alloy.	Slag.	Loss.	Alloy.	Slag.	Loss.
	Per ct.	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*	*Per ct.*
56a	37	49	14	34.5	11	53.5
56b	17	91	[a]8	34.5	32.5	33
56c	6	83	11	7.5	26 [b]41	52.5
56d	2	78	20	9	23 [b]38	53
56e	26	74	0	40	7.5	52.5
57a	1	97.5	1.5	2	24.5	72.5
57b	33	56	11	19.5	24.5	56

[a] Gain.

[b] Including dross.

The results of runs 56b, 56c, and 57a indicate that by proper care 85 to 90 per cent of the manganese could be recovered in the slag, with only about 25 to 30 per cent of the phosphorus, the slag having about 23 per cent manganese and 0.10 per cent phosphorus where the original ore had 24 and 0.37 per cent, respectively, or, on the basis of the calcined ore (deducting loss on ignition), 27.5 and 0.42 per cent.

Attempts to smelt the dephosphorized slags without the proper flux and with too little reducer gave low recoveries of manganese. After tests had indicated the flux and reducer needed, similar tests were again made. Not enough ore from lot 2b was left for a run, so ore from lot 4 was added. The charges were as follows:

In run 76 (dephosphorization), 57 pounds of lot 4 ore, averaging 25.4 per cent Mn, 2.1 per cent Fe, 37.2 per cent SiO_2, and 7.6 per cent $CaO+MgO$; 43 pounds of lot 2b ore, averaging 1.7 per cent Al_2O_3, 0.29 per cent P, and 12.1 per cent loss on ignition; 4 pounds of charcoal; and 7½ pounds of CaF_2, were charged in two batches. The results were as follows:

Results of run 76 (dephosphorization).

	Charge 1.	Charge 2.
Time	2 h. 10 m.	1 h. 35 m.
Kw.-hours used	125	100
Alloy made ... pounds	2.9	4.5
Manganese in alloy ... per cent	54.	64.1
Phosphorus in alloy ... do	2.5	2.12
Slag made ... pounds	96.7	82
Manganese in slag ... per cent	23.4	20.3
Phosphorus in slag ... do	.1	.09
Distribution of manganese:		
Metal ... per cent	9	
Slag ... do	79	
Loss ... do	12	
Distribution of phosphorus:		
Metal ... do	29.5	
Slag ... do	29.5	
Loss ... do	41	

The second tap was run a little too long and too much metal reduced.

The slags were then smelted. In order to determine what the results would be if both slags and metal were run into the smelting furnace from the dephosphorizing furnace, the metals as well as the slags were recharged.

In run 77 (smelting) the first charge consisted of 96.7 pounds of slag, 2.9 pounds of metal from run 76 (1), 17 pounds of charcoal (charcoal only was used, the coke supply being exhausted), and 7½ pounds CaF_2. The second charge consisted of 82 pounds of slag, 4.5 pounds of metal from run 76 (2), 17 pounds of charcoal, and 7½ pounds CaF_2. The results were as follows:

Results of run 77 (smelting).

	Charge 1.	Charge 2.	Total (or average.)
Time	2 h. 35 m.	1 h. 55m.	4 h. 30 m.
Kw.-hours used	175	130	305
Metal recovered ... pounds	23. 7	21. 5	45. 2
Manganese in metal ... per cent	75. 2	72. 5	74
Silicon in metal ... do	......	......	17. 2
Phosphorus in metal ... do	. 53	. 53	. 53
Slag recovered ... pounds	50	66	116
Manganese in slag ... per cent	9. 8	8. 8	9. 2
Phosphorus in slag ... do	. 06	. 07	. 65

Based on the manganese and phosphorus in the slag and metal charged in run 77, the distribution was as follows:

Percentage distribution of Mn and P based on analysis of slag and metal charged.

	Mn.	P.
Metal	75	70. 5
Slag	24	20. 5
Loss	1	9

Based on the analysis of original ore charged in run 76, the distribution when both slag and metal of that run were recharged was as follows:

Percentage distribution of manganese and phosphorus, based on original ore.

	Mn.	P.
Final metal	66	41. 5
Final slag	21	12
Loss, dephosphorizing run	12	41
Loss, smelting run	1	5. 5

The fact that the manganese loss was 12 per cent in the first (dephosphorizing) run, as compared with 1 per cent in the second (smelting) run, although the temperature was much higher in the second, indicates that the loss in smelting is more by dusting than by volatilization. The true volatilization loss appears to be small as long as the furnace is run as a resistance furnace, or with only a small submerged arc. A large, open, high-voltage arc would doubtless volatilize manganese.

To find what the results would have been had the metal obtained in run 76 not been recharged in run 77, we may subtract the 7.4 pounds of alloy, with its content of 4.4 pounds of manganese and 0.17 pound of phosphorus, from the 45.2 pounds of alloy obtained in run 77, with its 33.4 pounds of manganese and 0.24 pound of phosphorus. This would leave 37.8 pounds of alloy, containing 29 pounds of manganese, 7.75 pounds of silicon, and 0.07 pound of phosphorus, and analyzing about 77 per cent of manganese, 20 per cent silicon, and 0.18 per cent phosphorus, most of the iron being taken out in the first alloy.

The distribution, based on the content in the original ore, would then have been:

Percentage distribution of Mn and P when the alloy is not resmelted.

	Mn.	P.
Final metal	57	12
Final slag	21	12
Loss, dephosphorizing run	12	41
Loss, smelting run	1	5.5
Discarded in first metal	9	29.5

If the second charge in heat 76 had not been run so long, so that only 2 or 3 pounds of metal instead of 4½ pounds had been collected, the recovery, based on the original ore, would have been about 60 per cent.

However, if the high phosphorus alloy charged in run 77 had been left out, more of the phosphorus left in the dephosphorized slag might have gone into the metal, as the phosphorus may be expected to divide itself between the two liquid layers somewhat according to the concentration of phosphorus in each, the recovery of the 0.075 pound of phosphorus left in the two slags might have been practically complete. This would have given, in 37.8 pounds of alloy, 0.20 per cent phosphorus.

Two more slags were made from the same charges used in run 76, but the reduction was carried further, more metal being reduced than in run 76. The slags contained an average of 20 per cent manganese and 0.10 per cent phosphorus. These were smelted without the addition of the metal obtained in the dephosphorizing run and gave an alloy of about 72 per cent manganese, 27 per cent silicon, and 0.17 per cent phosphorus, with a recovery of 71 per cent manganese, based on the manganese content in the slags charged.

As in runs 56 and 57, the slags from an ore containing 0.37 per cent phosphorus were brought down to 0.10 per cent phosphorus in the dephosphorizing run, and as in run 76 an ore containing 0.29 per cent phosphorus came down also to 0.10 per cent phosphorus, it is probable that the slag of ores containing even more than 0.37 per cent phosphorus could also be brought down to about the same phosphorus content without much greater loss of manganese.

In order to get an idea of the power that would be needed in smelting the hot dephosphorized slag, the power consumption was noted in run 77 from the time the slag was thoroughly melted—that is, in about the condition it would be when tapped from the dephosphorizing furnace till the end of the heat. On the first heat, starting with a cold furnace, 95 kw.-hours was used and on the second 85 kw.-hours was used. On a third heat 70 kw.-hours would probably suffice.

The dephosphorization heats in run 76 were doubtless continued longer than necessary, as in heats c and d, run 56, and only 60 kw.-hours was needed.

CONCLUSIONS AS TO TWO-STAGE PROCESS.

It would appear that with a 3,000-kw. installation, using a 1,350-kw. furnace for dephosphorization and a 1,650-kw. furnace for smelting, 12,500 pounds of ore (containing 24 per cent Mn, 1.5 per cent Fe, 38 per cent SiO_2, 7.5 per cent CaO + MgO, 0.4 per cent P, and 1.5 per cent Al_2O_3, with 13 per cent loss on ignition), mixed with 250 pounds of coke, 250 pounds of charcoal, and 940 pounds of fluorspar (CaF_2), and given 5,500 kw.-hours, would produce 250 pounds of high phosphorus alloy, with about 50 per cent Mn and 2.5 per cent P, containing about half of the Fe in the ore and about 11,000 pounds of slag, with 23 per cent Mn and 0.10 per cent P. The latter tapped hot into the second furnace and mixed with 950 pounds of coke, 950 pounds of charcoal, and 860 pounds of fluorspar, and smelted with 7,000 kw.-hours, should produce 2,350 pounds (equivalent to a long ton of 80 per cent alloy) of an alloy containing 76 per cent Mn, 4.5 per cent Fe, 18.3 per cent Si, 1.0 per cent C, 0.20 per cent P, with a recovery of 60 per cent of the metallic manganese in the original ore.

The ore smelted direct without dephosphorization would give an alloy containing about 73 per cent Mn, 8 per cent Fe, 17 per cent Si, 1.25 per cent C, and 0.75 per cent P, with a recovery of 70 per cent of the manganese. A 3,000 k. v. a. installation including two furnaces as suggested above would produce daily 4.5 long tons of the dephosphorized alloy, containing 7,700 pounds of Mn, 1,850 pounds of Si, 456 pounds of Fe, 100 pounds of C, and 20 pounds of P, while a single 3,000 k. v. a. furnace would produce daily 5.5 long tons of the high-phosphorus alloy, containing 9,000 pounds of Mn, 2,100 pounds of Si, 950 pounds of Fe, 185 pounds of C, and 92 pounds of P.

Whether the lower phosphorus content resulting from the two furnace treatments would compensate for the lower manganese recovery and the lower output depends on whether the high-phosphorus alloy can be utilized.

If 0.9 per cent of standard 80 per cent ferromanganese—that is to say, 0.72 per cent ferromanganese—is added to the average steel bath, a content of 0.4 per cent phosphorus in the ferromanganese (the ratio of P to Mn being 1 to 200) would raise the phosphorus content of the steel by 0.0036 per cent. Ferro alloy of this ratio has been accepted by some steel makers during the manganese shortage.

The alloy containing 73 per cent manganese, 17 per cent silicon, and 0.75 per cent phosphorus would raise the phosphorus content of the steel by 0.0074 per cent, while the alloy containing 76 per cent manganese, 18.3 per cent silicon, and 0.2 per cent phosphorus would raise it only 0.0019 per cent. Either alloy would raise the silicon content of the steel 17 points, or a little less, as some silicon will be lost by oxidation.

An indefinite quantity of manganese alloys high in phosphorus can doubtless be absorbed by the steel industry. Cromlish,[a] in fact, suggested making a spiegel with 13 to 25 per cent manganese and 2.2 to 3.4 per cent phosphorus from "flush" and "tapping" cinders and utilizing this in the manufacture of the steel sheet, where a phosphorus content higher than normal is required to keep the sheets from sticking together in pack-rolling.

During the preliminary runs, and in some runs not reported herein, with ores high in phosphorus, when the proper flux or reducer was not used, a number of alloys running from 65 to 75 per cent Mn, 3 to 17 per cent Si, 1.3 to 5.3 per cent C, and 1.6 to 0.7 per cent P were produced. An assortment of these alloys has been sent to the Minneapolis station of the Bureau of Mines, where it is planned to study the possibility of eliminating the phosphorus.

According to Lang,[b] by melting ferromanganese high in phosphorus under manganese oxide (MnO_2) at 1,200° C., the phosphorus may be eliminated. A couple of tests were made at Ithaca by melting the ferro alloys high in phosphorus under high-grade manganese-oxide ores. No dephosphorization of the alloy, but instead a dephosphorization of the ore occurred.

TESTS WITH HIGH-SILICA ORES.

An important class of low-grade manganese ores is that in which manganese and iron occur in about equal amounts, with or without a high proportion of silica or phosphorus, or both, to manganese.

---

[a] Cromlish, A. L., U. S. Patent 1261907, Apr. 9, 1918.

[b] Lang, G., German Patent 252166, class 18 b, group 2, Oct. 14, 1912.

Many Cuyuna ores, such as the products of the Ferro and Mille Lacs mines, some of the Appalachian ores, and some ores from Leadville, Colo., fall into this class.[a]

A small lot of such an ore was at hand from the Sultana mine, Cuyuna district. The lump ore analyzed 29.0 per cent Mn, 24.2 per cent Fe, 4.5 per cent SiO_2, 1.5 per cent CaO + MgO, 3.7 per cent Al_2O_3, 0.20 per cent P, and showed 11.6 per cent loss on ignition.

Such an ore, smelted direct, would give about a 50 per cent manganese alloy. The ratio of phosphorus to manganese is high. It seems desirable to produce manganese alloys either of the composition of spiegel, which will bear cupola melting, for adding hot without excessive loss of manganese; or of that of 70 to 80 per cent ferromanganese for adding cold. Alloys of intermediate composition, say 40 to 60 per cent manganese, where the balance is iron instead of silicon, will chill the steel bath badly if added cold on account of the large amounts of inert iron carried. In order to add such alloys hot, electric melting of the alloys is necessary, as they would lose too much manganese in cupola melting. Electric melting, even of 80 per cent ferromanganese, is probably desirable, but few American steel plants are doing so as yet, and the necessity of installing electric furnaces in order to utilize the 50 per cent manganese alloys would be an obstacle to the adaption of electric melting. Where silicomanganese containing 60 per cent manganese and 20 per cent silicon is used instead of both ferromanganese and ferrosilicon, the inert iron carried by the silicomanganese is less than that in an equivalent mixture of 80 per cent ferromanganese and 50 per cent ferrosilicon.

APPLICABILITY OF TWO-STAGE PROCESS TO CUYUNA ORES.

As so many Cuyuna ores carry an undesirably high proportion of phosphorus to manganese, it appears that if electric smelting is used, a differential, two-stage, two-furnace process is worth consideration. The bulk of the iron, a little manganese, and much of the phosphorus may be reduced in the first furnace to a high-phosphorus, manganiferous pig iron, or, in an ore sufficiently low in phosphorus, to a spiegel, leaving in both cases a slag high in manganese, low in iron, and low in phosphorus to be tapped hot into the second furnace and there smelted to ferromanganese.

RESULTS OF TESTS.

Tests with the two-stage process were made with Cuyuna ores. In run 54, three heats were made, each charge including 100 pounds of Sultana lump ore, 3¾ pounds limestone (the furnace bottom had been patched with dolomite before this run, and some of the dolomite went into the slags), and 12 pounds of charcoal (except in heat c, where 10 pounds was used). The results were as follows:

---

[a] Harder, E. C., Manganiferous iron ores: Bull. 690, U. S. Geol. Survey, 1918, pp. 5, 8, 11.

Results of run 54.

Heat.	Time.		Kw.-hours used.	Alloy.	Analysis of alloy.					Slag.	Analysis of slag.		
					Mn.	Fe.	Si.	C.	P.		Mn.	Fe.	P.
	H.	*min.*		*Lbs.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*Lbs.*	*P. ct.*	*P. ct.*	*P. ct.*
a	2	15	100	30.2	24.5	68.1	0.07	2.35	0.17	60.0	32.0	2.7	0.04
b	1	20	58	20.3	11.4	82.5	.23	1.7	.19	41.0	44.8	5.9	.06
c	1	20	50	20.0	10.5	84.0	.02	2.25	.19	51.0	46.9	4.4	.03
d[a]	0	15	10	13.0	17.1	75.3	.40	2.10	.21	21.0 [b]15.5	36.1 26.1	3.0 13.8	Tr. .08

DISTRIBUTION OF Mn Fe, AND P.

	Mn.	Fe.	P.
	Per cent.	*Per cent.*	*Per cent.*
In metals	13.5	75	22
In slags	75.5	12	13
Volatilization and dust loss	11.0	13	65

[a] A little CaF_2 was added and a longer run was made to get the slightly stiff slag to tap more cleanly from the furnace.

[b] Mixed slag and metal shot taken from furnace bottom when cold.

The reduction in heat a was carried too far, reducing too much Mn. With experience it should be possible to leave about 80 per cent of the Mn, 15 per cent of the Fe, and 20 per cent of the P in the slag; take out 10 per cent of the Mn, 75 per cent of the Fe, and 20 per cent of the P in the metal; and lose 10 per cent Mn, 10 per cent Fe, 60 per cent P by volatilization and dusting.

Two thousand pounds of ore plus 200 pounds of reducer (half charcoal, half coke) plus 200 pounds of limestone, smelted with 1,000 kw.-hours or less, would then produce about 425 pounds of alloy (low-grade spiegel) containing 13 per cent Mn, 89.5 per cent Fe, 2 per cent C, and 0. 20 per cent P; and 1,100 pounds of slag containing 42 per cent Mn, 6.5 per cent Fe, 8 per cent SiO_2, and 0.07 per cent P, that is the equivalent of a high-grade ore.

This slag tapped hot into the second furnace, with the addition of 50 pounds of limestone and 330 pounds of reducer (half coke, half charcoal), and smelted with 700 kw.-hours, should give, assuming an 80 per cent recovery of the Mn in the slag, 460 pounds of alloy, analyzing 78.5 per cent Mn, 14.4 per cent Fe, 1.0 per cent Si, 6.0 per cent C, and 0.15 per cent P, and about 550 pounds of waste slag.

The recovery of the Mn in the final alloy would be 64 per cent of that in the original ore. A 3,000-kw. installation would then use a 1,750-kw. furnace for the first operation and one of 1,250-kw. for the second.

In run 55, 190 pounds of slag chosen from the slags made in run 54 and in another similar run were mixed, the composition of the mixed slags being 41.0 per cent Mn, 15 per cent SiO_2, 3.4 per cent Fe, and 0.06 per cent P.

Results of run 55.

Heat.	Charge.		Charcoal.	Coke.	Mill scale.	Steel scrap.
	Mixed slags.	$CaCO_3$				
	Pounds.	*Pounds.*	*Pounds.*	*Pounds.*	*Pounds.*	*Pounds.*
a	100	10	18	....	2.5	....
b	90	9	8	8	....	1.5

Heat.	Time.	Kw.-hours used.	Alloy.	Analysis of alloy.			Slag.	Analysis of slag.	
				Mn.	Si.	P.		Mn.	P.
	H. min.		*Lbs.*	*Per ct.*	*Per ct.*	*Per ct.*	*Lbs.*	*Per ct.*	*Per ct.*
a	2 50	176	43.5	74.2	3.2	0.09	48.2	11.8	(a)
b	2 10	106	41.0	74.5	5.7	.10	45.0	5.4	(a)
Total	5 0	282	84.5	......	......	......	83.2	......	......
Average	......	......	......	74.3	4.4	.10	......	8.7	......

DISTRIBUTION.

	Mn.	P.
	Per cent.	*Per cent.*
Metal	82½	75
Slag	6½	....
Loss	11	....

a Not determined.

The results of this run checked the assumptions made as to recovery and power comsumption in smelting the slag.

If the ore were high in phosphorus, the first furnace would be so operated as to throw as little manganese into the alloy as possible, and produce a pig iron with a manganese content of about 10 per cent and a phosphorus content according to that in the ore. Such pig iron would doubtless find use when mixed with other pig of suitable composition and should have some value. If the ore is low enough in phosphorus, a little more reducer would be used in the first furnace so as to cause enough manganese to be thrown down to form a standard spiegel. If too little iron in proportion to the manganese was left in the slag for the second smelting, steel scrap or ore low in manganese but high in iron would be added, so that the products from a low phosphorus ore could be speigeleisen and ferromanganese.

COMPILATION OF DATA.

The probable performance of several types of domestic manganese ore under electric smelting, compiled from the data available on present practice, and from the results of experimental work, is given in Table 31 following.

The electrode consumption per ton of product has been calculated on the basis of kw.-hours required per ton of product, taking as standard the normal commercial consumption of 175 pounds of electrode per ton of 80 per cent ferromanganese produced from an ore containing 40 per cent manganese and 20 per cent silica, and using 5,500 kw.-hours. In other words, the electrode consumption per 1,000 kw.-hours used is 31 pounds. The actual consumption of electrodes, not including stub ends in the experimental runs, varied from 6 to 10 pounds per 450 kw.-hours used, which is of the same general order of magnitude as the figures assumed.

In Table 32 an attempt has been made to calculate costs of operation for the eight ores represented in Table 31, on the assumption that the ores are priced as follows: Ores 1 and 2 in accordance with the War Industries Board's schedule, ores 3 and 4 at $15 per ton, ore 5 at $7.50 per ton, ores 6 and 7 at $7 per ton, and ore 8 at $8 per ton. Limestone is assumed to cost 0.2 cent a pound, fluorspar 3 cents a pound, coke 0.5 cent a pound, charcoal 1½ cents a pound, steel scrap 2 cents a pound, electrodes 15 cents a pound, and power 0.5 cent per kw.-hour used. Selling prices are calculated at $285.80 per long ton of 80 per cent ferromanganese (equivalent to $250 per long ton of 70 per cent ferromanganese), and $150 per long ton of 50 per cent ferrosilicon; and the manganese and silicon contents in a silicomanganese alloy are calculated as having the same value as in the separate ferro alloys. The alloy of ore 5, with 0.60 per cent phosphorus, is figured as worth $260 per equivalent ton of 80 per cent ferro, while that of ore 7, with 0.75 per cent phosphorus, is figured as worth $235 per equivalent ton. All these figures are assumed, and would vary widely according to the situation of the plant. They are included merely to give a rough idea of the costs. Peace-time figures would greatly alter the costs and profits.

Lyon, Keeney, and Cullen[a] give the following prices per metric ton in Germany, f. o. b. Louisberg, January 1, 1913:

50 per cent ferrosilicon	$77.55
80 per cent ferromanganese	68.00
Silicomanganese (68 to 75 per cent Mn, 20 to 25 per cent Si)	106.50

It is seen that the price paid for contained manganese and silicon in the silicomanganese is higher than that paid for the same amount of manganese and silicon in ferro alloys.

---

[a] Lyon, D. A., Keeney, R. M., and Cullen, J. F., work cited, p. 140.

The greater daily output per furnace when using the higher grade ores makes their use more profitable to the electric-smelting plant. On the assumptions made, the practice of smelting ores that can be run without a slag to silicomanganese shows the largest profit as well as the highest recovery of manganese from the ore.

The figures for ores 1 to 4 have been calculated on the basis of commercial practice, and for ores 5 to 8 on the basis of the experimental work in which expensive fluorspar and charcoal was used. Experience in large-scale operation would probably allow the substitution of some limestone for fluorspar and some coke for charcoal. No experiments to test this possibility were made, as the work was aimed to show the metallurgical rather than economic possibilities, and as such tests on a laboratory scale would have to be checked by large-scale tests before accurate information would be obtained.

According to Willcox,[a] Germany made silicomanganese from blast-furnace slags, and ferro or silico from low-grade ores during her manganese shortage.

The Taylor-Wharton Steel Co., High Bridge, N. J., which makes manganese steel, had a slag from manganese steel running 35 to 42 per cent silica and 31 to 43 per cent manganese.[b] The company had some of this slag electrically smelted into an alloy of 47 per cent manganese and 20 per cent silicon, which was used successfully in the manufacture of steel. The cost of smelting was too high to make the use of the alloy economically desirable at the peace-time prices of ferro then prevailing.

Tables 31 and 32 follow on next pages.

CONCLUSION AS TO ELECTRIC SMELTING.

The conclusion to be drawn from this investigation is that although the electric smelting of manganiferous slags and low-grade domestic ores is unlikely to be profitable at times of normal costs and prices, such smelting is metallurgically possible, and could be done profitably in times of high prices.

---

[a] Willcox, F. H., The significance of manganese in American steel metallurgy: Trans. Am. Inst. Min. Eng. Vol. 56, February, 1917, pp. 412–420.

[b] Personal communication.

TABLE 30.—*Data showing probable results of electric smelting of domestic manganese ores.*

[Based on results of experiments and on data available from present practice.]

Ore No.	Kind of ore.	Results of electric smelting of domestic manganese ore.								Analysis of alloy made.					Analysis of slag made.		
		Analysis of ore.						Loss on ignition.	Recovery of Mn in ore as alloy.	Mn.	Fe.	Si.	C.	P.	Mn.	CaO.	SiO₂.
		Mn.	Fe.	SiO₂.	CaO+MgO.	Al₂O₃	P.										
		P. ct.	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*	*P. ct.*
1	Mixed Colorado and Utah	40.0	1.0	13.0	5.0	5.0	0.15	13.0	75	80.0	12.8	1.0	6.0	0.20	12	35	30
2	Washed Philipsburg	42.0	1.3	20.0	6.5	4.0	.10	12.0	72	80.0	11.8	2.0	6.0	.18	......	......	30
3	Mixed California ore plus quartz	30.0	1.0	35.0	.5	.5	.11	12.0	95	53.0	23.8	23.0	.6	.18	......	......	......
4	Same as No. 3 without steel scrap	30.0	1.0	35.0	.5	.5	.11	12.0	95	68.0	2.0	29.5	.3	.23	......	......	......
5	Philipsburg screenings	26.4	2.5	36.4	6.5	2.0	.21	14.5	70	71.0	11.4	15.5	1.5	.60	11	25	55
6	Philipsburg lump—one operation	24.0	1.6	38.0	7.5	1.6	.37	12.8	70	73.0	1.7	17.0	1.5	.75	12	25	55
7	Philipsburg lump—two operations	24.0	1.6	38.0	7.5	1.6	.37	12.8	24	50.0	41.0	.5	6.0	2.50	23	......	......
									60	76.0	4.5	18.3	1.0	.20	10	25	55
8	Sultana lump—two operations	29.0	24.2	4.5	1.5	.3	.15	12.3	10	13.0	84.5	.3	2.0	.20	42	......	......
									64	78.5	14.4	1.0	6.0	.15	10	40	30

Ore No.	Kind of ore.	Charges required to produce 1 long ton of 80 per cent ferro or its equivalent in Mn (1,792 pounds).						Slag made per 1,792 pounds of Mn	K.W.H. used per 1,792 pounds of Mn	K.W.H. per long ton of the alloy made.	Carbon electrodes used.		Output per day per 3,000 k.v.a. connected load; 90 per cent power factor, 90 per cent load factor assumed.		Silicon output per day, calculated to equivalent 50 per cent Si ferro-silicon.	Long tons of high-Mn high-P pig iron.
		Ore or mixture.	CaO_3.	CaF_2.	Coke.	Charcoal.	Steel scrap.				Per 1,792 pounds of Mn made.	Per long ton of alloy made.	Alloy.	Equivalent 80 per cent ferro.		
		Lbs.	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*	*Lbs.*			*Lbs.*	*Lbs.*	*Long tons.*	*Long tons.*	*Long tons.*	
1	Mixed Colorado and Utah	6,000	1,100	....	1,800	....	230	2,600	5,000	5,000	155	155	11.7	11.7	....	....
2	Washed Philipsburg	6,000	2,100	....	1,900	....	190	4,000	5,500	5,500	175	175	10.6	10.6	....	....
3	Mixed California ore plus quartz	6,300	....	....	2,400	....	700	....	8,750	5,800	270	180	10.0	6.6	5.8	....
4	Same as No. 3 without steel scrap	6,300	....	....	2,460	....	....	....	8,700	7,400	215	185	7.9	6.7	5.8	....
5	Philipsburg screenings	10,250	....	1,450	1,200	1,200	....	6,000	10,000	9,000	310	280	6.5	5.8	2.0	....
6	Philipsburg lump—one operation	10,700	....	1,500	1,200	1,200	....	6,200	11,500	10,500	355	325	5.5	5.0	1.9	....
7	Philipsburg lump—two operations	12,500	....	940	250	250	....	[b]11,000	[c]5,500	....	390	365	4.5	4.3	1.65	....
		[d]11,000	....	860	950	950	....	6,000	[d]7,000	....	....	....	....	....	....	7.1
8	Sultana lump—two operations			1,800	1,200	1,200			12,500	11,800						
		10,000	1,000	....	500	500	....	[b]5,500	[c]5,000	....	265	260	6.9	6.8	....	....
		[b]5,500	250	....	825	825	....	2,750	[f]3,500	....	....	....	....	....	....	....
			1,250		1,325	1,325			8,500	8,350						

[a] 250 pounds of waste metal. [b] Slag A. [c] In 1,350-k. w. furnace. [d] In 1,650-k. w. furnace. [e] In 1,750-k. w. furnace. [f] In 1,250-k. w. furnace.

TABLE 31.—*Data showing calculated costs of electric smelting of domestic manganese ores.*

Ore No. (See Table 31)	1	2	3	4	5	6	7	8
Kind of ore	Colorado and Utah.	Washed Phillipsburg.	California ore and quartz.	Same as 3 without steel scrap.	Philipsburg screenings.	Philipsburg lump—one operation.	Philipsburg lump—two operations.	Sultana lump—two operations.
Cost of ore per ton	$38.30	$36.25	$15.00		$7.50	$7.00	$7.00	$8.00
Cost per long ton of 80 per cent ferro or equivalent Mn (1,792 pounds):								
Ore	102.50	97.50	42.00	$42.00	34.30	34.40	39.00	35.80
$CaCO_3$	2.20	4.20						
CaF_2					43.50	45.00	54.00	
Coke	9.00	9.50	12.00	12.00	6.00	6.00	6.00	6.65
Charcoal					18.00	18.00	18.00	19.90
Steel scrap	4.60	3.80	14.00					
Power	25.00	27.50	43.75	43.50	50.00	57.50	62.50	42.50
Electrodes	23.15	26.20	40.50	40.50	46.50	54.50	58.50	40.00
Total	166.45	168.70	151.25	138.00	198.30	215.30	238.00	147.35
Value of product:								
On basis of long tons of 80 per cent ferromanganese at $285[a]	(1) 285.00	(1) 285.00	(1) 285.00	(1) 285.00	(1) [b]260.00	(1) [c]235.00	(1) 285.00	(1) 285.00
On basis of long tons of 50 per cent ferrosilicon at $150[a]			(0.88) 132.00	(0.88) 132.00	(0.35) 52.50	(0.38) 57.00	(0.38) 52.00	
On basis of long tons of pig iron with 10 per cent Mn at $30							1.05	31.50
Calculated value of alloy per 1,792 pounds of Mn contained	285.00	285.00	417.00	417.00	312.50	292.00	337.00	316.50
Difference-value of product less cost of raw materials	119.55	116.30	265.75	279.00	114.20	76.70	99.00	169.15
Difference times equivalent tons of ferromanganese made per day[a]	(11.7) 1,400.00	(10.6) 1,235.00	(6.6) 1,750.00	(6.7) 1,870.00	(5.8) 605.00	(5.0) 383.00	(4.3) 425.00	(6.8) 1,150.00
Assumed cost of superintendence, labor, overhead, interest, depreciation, etc., per day, per 3,000 k. v. a. of plant capacity	400.00	400.00	400.00	400.00	400.00	400.00	400.00	400.00
Calculated profit per day per 3,000 k. v. a.	1,000.00	835.00	1,350.00	1,470.00	205.00	(loss) 17.00	25.00	750.00

[a] Figures in parentheses represent tons.
[b] Assumed penalty for 0.60 per cent P, $25 per ton of product equivalent to 80 per cent ferro manganese.
[c] Assumed penalty for 0.75 per cent P, $50 per ton of product equivalent to 80 per cent ferro manganese.

CHAPTER 11.—USE OF MANGANESE ALLOYS IN OPEN-HEARTH STEEL PRACTICE.

By Samuel L. Hoyt.

INTRODUCTORY STATEMENT.

This report presents the results of an extensive study of the use of manganese alloys in open-hearth steel practice in the United States. The magnitude of the work and the number of men who aided in one way or another make difficult acknowledgment to all who have contributed to its progress. However, some expression should be made of the hearty cooperation accorded to the members of the bureau by the various manufacturing interests. With such a sympathetic attitude, the investigation, depending so largely, as it did, upon cooperation between the steel plants and the investigators, was early assured of every possibility of ultimate success. The war conditions that rendered this investigation necessary have passed, but it is hoped that the results may prove of permanent value to the open-hearth steel industry.

PURPOSE AND SCOPE OF INVESTIGATION.

The purpose of making this investigation was to determine the extent to which domestic or low-grade manganese alloys could properly be substituted in open-hearth steel practice for high-grade alloys without materially impairing the steel production either as to quality or quantity. Moreover, it was held that such an extensive investigation of this important step in the manufacture of steel would undoubtedly yield valuable results to the steel industry as well as contribute, in no small way, toward directing future investigations in the same field.

It was recognized at the start that data bearing on the projected study should be available at individual plants, and that a compilation and digest of such results would be the logical method of approaching the proposed investigation. This consideration somewhat controlled the selection of the steel plants at which to conduct the detailed investigations.

The evolving of a definite experimental program from the statement of the general problem, considering the time element and the many and varied factors involved, was not reached without due consideration of competent metallurgical advice. After making a preliminary survey, it seemed important to determine (*a*) the conditions in open-hearth practice that would lead to a conservation of manganese, both during the working of the heat and in making the final additions; (*b*) the most satisfactory metallurgical conditions for the use of manganese in the form of low-grade or special alloys; and (*c*) the effect on the finished steel, both as to quality and "condition," of the various methods and processes studied. With these points in mind, the selection of the steel plants was made so that research work bearing on one or more of these points could be conducted.

It was decided to determine slag and metal compositions during the refining of the heat; also, the temperature was to be noted each time a sample was taken, in order to determine, if possible, the temperature effect. The "recovery" of manganese was to be determined from the residual and final manganese contents and the weight of the metal. To this end a sample of the finished steel was taken during teeming. By taking three such samples, one at the beginning, one toward the middle, and one at the end of teeming, tests for uniformity were possible. This practice was generally observed throughout the investigation. The data obtained were also supplemented by the plant records covering given heats as well as by personal observation during refining, pouring, and teeming.

When planning steps that should be taken to determine the quality and "condition" of the steel, it was found that no definite and well-proved method was available. True, the open-hearth melter knows whether his heat is in proper condition, but what was needed was a quantitative estimate of "condition." Without attempting to discuss the physical chemistry of a heat of molten steel, it may be said that the condition of the heat must depend, aside from the temperature, upon the presence in the steel in those substances that affect the "condition." Of these there are two kinds: (1) Substances that promote "openness," or the gases, which again may be classified as (*a*) gases that are products of chemical reactions, being, in so far as we know, CO and possibly CO_2, and (*b*) gases that are absorbed from the furnace gases, such as H, N, CO, and CO_2; and (2) substances that promote "soundness," such as the reducing and solidifying agents, C, Mn, Si, and Al.

In general it is held that Mn, Si, and Al inhibit the chemical reactions producing CO by reducing (or partly reducing) FeO, the principal constituent that produces the reactions. In this statement only the metal bath is considered and the FeO, and not Fe_2O_3, is assumed to be in solution in the steel. According to this idea, reducing action on a slag containing Fe_2O_3 would produce FeO, part of which would enter the steel to react later with C, Mn, and other reducing agents present. The reduction of FeO, then, is the principal means of "settling" the liquid steel, and it is for this reason that Mn is added in the final steps. It is also held that Si and Al produce solidity in the finished steel, aside from reducing FeO and CO, either by keeping the gases H, N, etc., in solid solution, or by preventing the dissociation of the compounds of those elements and iron.

The obvious procedure to get a quantitative estimate of the "condition" of the steel, considering both the behavior of the molten metal and the character of the ingot, would be to determine the amounts of the constituents in each of these two groups and to weigh one set against the other. Even this procedure would not, at present, lead to results that could be interpreted with entire confidence, even though there were no uncertainties in the analytical methods, because we do not know the quantitative effect of each constituent, either by itself or when associated with other constituents in varying amounts. In view of this lack of fundamental data, it was decided to make the analyses and use the results in a qualitative way, at least, to compare the different practices investigated.

THE FUNCTIONS OF MANGANESE.

During such a critical period as that now passed, the question might be raised as to the possibility of eliminating manganese from steel making. This point was duly considered but it was at once held that the use of manganese is not merely an expedient, for which some substitute might readily be had, but is rather one of the basic requirements of successful practice in working steel. It is quite true that in many instances the actual amount of manganese used in a heat of steel is greater than purely metallurgical considerations demand, and any excess could well be considered as so much wasted.

It may be well to review briefly the important functions of manganese as they bear directly on both of the points mentioned above. The first function of manganese, broadly considered, is to refine and "settle" the molten bath of steel. The aim here is to put the metal in a proper condition for pouring, and to produce ingots (or castings) of the desired quality and texture. Manganese is not the most efficient element that can be used for this purpose, calculated from the heat of combustion of the element to its oxide, but is without doubt the most satisfactory because of the excellent condition (freedom from objectionable foreign inclusions) in which it leaves the bath.

The proportion of manganese theoretically required for this operation might possibly be calculated from the amount of oxygen converted from the active form, FeO, to the inactive form, MnO. Assuming an oxygen content of 0.075 per cent in the unsettled steel and of 0.015 per cent in the finished steel (oxygen by the Ledebur method), the amount of manganese used in this way would be 0.2

per cent. The writer is informed by the Bureau of Standards that such a calculation is premature, owing to lack of knowledge on the subject of "deoxidation" and the faultiness of the Ledebur determinations. However it would seem to the writer, from the work done at the Bureau of Standards,[a] that the amount of oxygen determined is the amount present as FeO (active form), subject possibly to an error due to partial reduction of CO during the determination. At any rate, the above is advanced as at least the first approximation of the amount of manganese required simply for destroying the ferrous oxide present in the bath. The amount of manganese required naturally would vary with the condition of the bath and, in order to insure efficient "deoxidation," would be somewhat in excess of the calculated amount. A well-made heat of steel would probably not require more than 0.35 per cent Mn.

Manganese is also desirable in steel to improve the rolling properties, in which capacity it appears to serve a dual purpose. First of all, manganese deoxidizes and refines the molten steel in such a way as to give ingots of the desired texture without robbing the steel of its hot-working properties. Thus, ingots may be rolled into finished shape, without the formation of excessive fissuring or surface defects. Other reducing agents, such as aluminum and silicon, are prone to leave the metal in poor condition for rolling and forging. They eliminate one cause of hot shortness—iron oxide—but fail to convert the sulphur into a harmless form, as manganese does, and leave behind their highly refractory oxides, both of which tend to produce poor rolling qualities. Secondly, manganese, by retarding the rate of coalescence or grain growth, renders steel less sensitive to the effects of the high temperatures used in rolling and is supposed to promote plasticity, at least in ordinary steels, at rolling temperatures. Silicon and aluminum, on the other hand, increase, rather than decrease, the grain size of steel. The proportion of manganese required in this capacity probably does not exceed 0.35 per cent in well-made steel.

Finally, manganese is desired in the finished steel to produce certain physical or mechanical properties or to make the steel more amenable to subsequent heat treatment.

The foregoing discussion indicates that manganese is an important factor in the steel industry. Of course, material such as "American ingot iron" can be successfully rolled, even though no manganese be added, but requires greater time and care.

It is of interest to note that manganese, coming in the periodic system between iron and the strengthening elements on one side and the hardening elements on the other, has the dual function of strengthening and hardening steel, which is not possessed by any other element.

---

[a] Cain, J. R., and Pettijohn, Earl. A critical study of the Ledebur method for determining oxygen in iron and steel: Bureau of Standards Technologic Paper 118, 1919, 33 pp.

Manganese conservation would best be obtained by eliminating the manganese specification except where the amount of manganese present in the finished steel has some definite bearing on the properties of heat treatment of the steel. In other words, whenever only casting and rolling-mill practice (plant problems) are involved, the steel man should be allowed to exercise his own judgment as to the amount of manganese that should be used to give the most satisfactory and economical practice, and the finish and quality of the product should be controlled by adequate inspection. On the basis of Ellicott's figures,[a] by reducing the manganese requirements by 0.2 per cent in making plates and shapes and other low-carbon steel (estimated production, 21,350,000 tons), 54,000 tons of 80 per cent ferromanganese could be saved.

RECOMMENDATIONS FOR THE UTILIZATION OF DOMESTIC ALLOYS.

The investigation here reported indicated that three practices for utilizing domestic alloys in open-hearth steel practice seem to commend themselves above the others. These are as follows (but not in the order of their importance): (1) The use of a "molten spiegel mixture" for deoxidation and recarburization; (2) the practice of melting and refining the steel bath so as to insure a comparatively high residual manganese content, say, 0.3 per cent Mn; (3) the use of manganese alloys containing silicon. In selecting plants for investigating these practices two points were kept in mind. The plant should have either "ordinary" practice, for the sake of comparison, or else one of the three just mentioned, and the product or kind of steel made should be representative of the larger tonnages, such as shell steel, plates, or sections.

"MOLTEN SPIEGEL MIXTURE" PRACTICE.

The practice has been adopted at a few plants of combining in one operation both recarburization and deoxidation by using a mixture of pig iron and spiegel that has been premelted in a cupola. This "molten spiegel mixture" contains 5 to 11 per cent metallic manganese, 4 per cent carbon, and the desired amount of silicon, and is added to the ladle during the tapping in such a way as to cause a thorough and uniform mixture of the two streams.

The principal advantages of interest here, not considering questions of plant and operating economy, are as follows: 1. A low-grade or domestic alloy can be used in the preparation of the "mixture."

---

[a] Ellicott, C. R. Manganese conservation in steel making: Iron Age, vol. 101, June 6, 1918, pp. 1484–1485.

2. The deoxidation is accomplished by means of a dilute solution, with a consequent increase (on theoretical grounds) of the efficiency of the deoxidizer. This point will receive further consideration. 3. The deoxidizer is added in the molten state, insuring certain attendant advantages, which will also be considered at greater length. 4. A special advantage, if a large steel output is desired, is that the amount of the recarburizer is comparatively large and the capacity of the plant is materially (and economically) increased thereby. There is some question as to the propriety of including this advantage as peculiar to this particular practice. The use of pig iron as a recarburizer may be accomplished in other ways with the same economy and increase in plant capacity. 5. Another advantage would seem to the writer to be as follows: As compared with the results in the usual practice of adding carbon and manganese, there should be less likelihood of missing a heat.

This practice, at least at the plant visited, and it is understood to be the same elsewhere, is limited to the manufacture of the high-carbon steels or those running 0.30 per cent or more of carbon. To make steels with 0.20 per cent carbon would require working the bath until the carbon content was about 0.10 per cent, and the molten mixture added would have to contain about 20 per cent manganese (spiegel). The amount of the addition would be reduced from 13,000 pounds to about 4,000 pounds, which would mean that some of the advantages just enumerated would be lessened, and, with the increased loss of manganese the practice would probably not be commercially feasible. However, when the other alternative—the use of ferromanganese, either solid or liquid—is considered, the practice of premelting spiegel in the cupola seems commendable, on grounds to be considered later. In the event of undue shortage of high-grade ferromanganese the practice would doubtless offer a ready solution of the problem of using domestic alloys in making steel for shapes, plates, etc. Against the increased cost of production, as compared with the cost of cold ferromanganese practice, there would be the greater uniformity of product and more uniform practice as an offset.

HIGH RESIDUAL MANGANESE PRACTICE.

At certain plants the practice of preferential oxidation and elimination of carbon and phosphorus has been developed, the residual manganese being kept at a comparatively high value, say, 0.25 to 0.30 per cent, as compared with 0.10 per cent manganese for a final carbon content of 0.10 per cent. This is accomplished, broadly speaking, (*a*) by rapidly removing the phosphorus and retaining it as stable calcium phosphate during the earlier and colder period of melting; (*b*) by maintaining a high finishing temperature and working the charge with a high manganese content so that the slag contains about 8 per cent manganese; and (*c*) by increasing the lime content of the slag to about 47 per cent as a minimum.

This process possesses undoubted advantages, but they are such that they are probably best appreciated by plants in which the process has been developed and where it is now in operation on a sound commercial basis. First of all it may be stated that the practice, correctly applied, leads to the production of high-grade and uniform steel, which in itself means increased rolling-mill output, fewer rejections, and a more ready market. This is largely due to the fact that the steel is made—where it should be made—in the furnace.

A second advantage derived from the high MnO and CaO contents of the slag, is that the final additions of manganese can be added in the furnace, with a recovery comparing favorably with that of ladle additions. A third advantage is that the same pig iron used for the charge, and containing appreciably more manganese than ordinary basic iron does, can be used to recarburize and partly deoxidize the bath. The rest of the manganese is added as ferromanganese. At a steel plant which operates in conjunction with a blast-furnace plant a harmonious and economical cycle of plant operations is made possible. At the same time the open-hearth slag can be resmelted in the blast furnace for the recovery of the iron and manganese and the utilization of the lime.

This practice is largely dependent upon the amount of phosphorus in the slag, for obviously it would not be worth while to recover the manganese at the expense of unduly increasing the phosphorus content of the pig iron. In this country we are fortunately situated in this respect, as there is still a large amount of ore rather low in phosphorus available. No definite figure can be given at this time as to the maximum allowable phosphorus content of the pig iron, but it is the opinion of at least one steel man who uses this process that a content of 0.6 per cent would not be excessive. Under the conditions prevailing in 1918, this practice had the additional advantages that the high-manganese pig iron could be procured by smelting domestic manganiferous iron ore and that the manganese alloy added to the furnace at the end of the heat could as well be spiegel as ferromanganese, assuming that the finished steel contains more than about 0.10 per cent carbon. There would also be certain disadvantages, particularly that the carbon content of the bath would have to be worked to a lower figure than in present practice. On account of the high cost of spiegel and the greater time required, it is doubtful whether the steel plants would substitute spiegel for ferromanganese. Another interesting point, as regards the utilizing of domestic manganiferous iron ore, is that low-silica ore could be added to the slag as a source of manganese oxide.

The high manganese content of the charge is generally obtained by using a "high-manganese" pig iron (2 to 3 per cent manganese), but may also be obtained by adding manganese ore to the slag or manganese alloys to the bath or by a combination of these methods. This point would be determined by plant economy, but it seems doubtful whether the practice would be worth while unless a high-manganese pig iron were available. The writer is informed by one blast-furnace superintendent that running the manganese up to 2 per cent does not materially affect the production, so that lower pig-iron production would not be a disadvantage in this practice. The loss of manganese by oxidation and transferrence to the slag is considerable. This loss may be kept at a minimum by increasing the basicity of the slag in CaO and FeO, which, combined with the MnO, which also acts as a base, exert the desired effect upon the manganese of the bath.

As the working of the charge progresses its temperature rises until finally with the high CaO, and particularly the high MnO content of the slag, the carbon is eliminated more rapidly than the manganese, with the result already stated, namely, the manganese can be held to about 0.3 per cent at the end of the heat. Present data indicate, unfortunately, that no material decrease in the amount of manganese required and no material increase in the recovery of manganese in the additions may be expected, so that the advantages are derived not from a decreased consumption, but from the form in which it can be added.

Data for one such heat showed that a total of 3,728 pounds of manganese was used in one form or another to produce 1,272 pounds of manganese in the finished steel—that is, 3.54 pounds was used to produce 1 pound in the steel. The manganese added in the recarburizer and as ferromanganese amounted to 1,068 pounds, of which, assuming the manganese loss to come from these two sources, 838 pounds was recovered in the finished steel, a recovery of 78.4 per cent. In this heat the ferromanganese was added to the furnace. Another more or less comparable heat selected at random, but more representative of "standard" practice, used 2,190 pounds of manganese to produce 1,200 pounds, or 1.82 pounds (as compared to 3.54 pounds) to produce 1 pound of manganese in the finished steel.

STANDARD OPEN-HEARTH PRACTICE COMPARED WITH CERTAIN OTHER PRACTICES.

Data regarding the results obtained in standard open-hearth steel practice as compared with the results when molten spiegel is used or when there is a large percentage of residual manganese will be presented in a later report after more complete analytical results have been received.

USE OF MANGANESE-SILICON ALLOYS.

The high silica content of most of our domestic manganese and manganiferous iron ores made it advisable to consider the possible use of manganese-silicon alloys in steel making in both acid and basic practice. For the purposes of the present discussion these alloys will be divided roughly into two classes—high-grade silicomanganese containing about 50 per cent manganese and 25 per cent silicon and low-grade silicospiegel with about 15 to 20 per cent silicon and 30 to 35 per cent manganese with 50 per cent iron. The manganese-silicon ratio of the first alloy is about 2 and of the second alloy $2\frac{1}{3}$ to $1\frac{1}{2}$. Each of these alloys would be made from the silicious manganese ores of California and Montana, and the low-grade alloys from the silicious manganiferous iron ores of Minnesota.

While there is nothing new about the practice of using manganese-silicon alloys in steel making, it may be well to review some of the points connected therewith.

It is understood that silicomanganese has been used fairly extensively in Europe, and in this country it was used at certain plants as standard practice until the supply was cut off by the war. Silicon is always an efficient reducing or settling agent when used in the customary small amounts, but it may or may not be desirable in the finished steel. On this account the possibility of using manganese-silicon alloys depends upon the amount of silicon that can be tolerated in the finished steel in the ingot form. In certain grades of steel, particularly in steel that must be welded, silicon should be low or practically absent. In steel for sheets and plates, which must give a good finished surface, the most efficient rolling-mill practice requires that the silicon be kept tolerably low; but it is believed that 0.10 to 0.15 per cent could be used, provided the manganese content were not too high. In forging steel, high-carbon steels, and castings, where the aim is to produce sound steel, more silicon can be used, or between 0.20 and 0.35 per cent. Of these three fields the latter is the one in which manganese-silicon alloys will find their first application. In the second field it seems quite probable that conditions (to be discussed later) will many times permit their use; but from the nature of things manganese-silicon alloys can not be used to make steels of the first group—those that must be welded.

ACID PRACTICE.

It is with considerable hesitation that the discussion of manganese-silicon alloys in open-hearth practice is approached, particularly as the controversial character of many of the points is so clearly recognized. Consequently, it may be well, at the outset, to state briefly the manner in which the writer first became interested in the possibilities of their use. A number of years ago the writer was conducting a series of experiments on the occurrence and identification of foreign inclusions in acid open-hearth steel, principally ordnance steel. In this work ferromanganese, ferrosilicon, and a mixture of ferromanganese and ferrosilicon were added to a steel sample taken shortly after "oreing"—that is, to "wild" steel—in an attempt to produce an excess of the constituent, or constituents, supposed to form as a result of the addition.

It seemed fairly clear as a result of this work that the use of silicon was apt to be dangerous, not on account of any harmful effect of the residual metallic silicon but because it produced a constituent (assumed to be SiO_2 or at least a highly refractory silicate) that was likely to remain in the ingot and produce hot shortness. Hence the idea was suggested that a manganese-silicon alloy might, and probably would, form a manganese silicate containing some ferrous oxide (a true slag) which would be fluid and would more readily coalesce into larger particles than SiO_2 would, and therefore free itself more readily from the steel. By using such an alloy it would then be possible to take full advantage of the use of silicon as a deoxidizer without suffering the usual attendant disadvantages of its use. None of the manganese-silicon alloy was available at the time so a parallel experiment could not be conducted.

As binary alloys are known to be generally more active, or powerful, than the weighted sum of the two constituents would indicate, it was also assumed that, aside from the possibility of obtaining a better separation of the insoluble products of the deoxidation process, the alloy of manganese and silicon would prove to be a more powerful reducing agent than ferromanganese and ferrosilicon used separately. On reflection, the thought occurs that manganese and silicon, reacting separately with FeO, would produce the oxides MnO and FeO or a silicate of iron. Manganese and silicon reacting as an alloy with FeO would produce a silicate of manganese, which may or may not form a double silicate with FeO. In either case we would expect to find the advantage in favor of the manganese-silicon alloy.

The relative weights of the silicomanganese and of the mixture of the ferromanganese and ferrosilicon will be considered at another place.

Another point of great technical importance is the percentage recovery of manganese when added as silicomanganese and as ferromanganese along with ferrosilicon. It should be stated that a 100 per cent recovery, based on the present theory of "deoxidation," is hardly possible, nor is it desirable. Such recovery would mean retention of the deoxidation products, to be determined later as metallic manganese and silicon. A method of addition that would lead to satisfactory deoxidation and yet would eliminate the loss due to admixture with the slag, volatilization, etc., and could be accomplished with the minimum amount of manganese, would be very desirable because it would lead to both conservation of manganese and uniformity of composition of the steel. Conservation of manganese would be given by the actual percentage recovery, and uniformity of composition would be assumed by the constancy of the percentage recovery.

Fortunately the writer was able to examine records of heats made with silicomanganese covering a period of several years, from which some fairly satisfactory conclusions may be drawn bearing on these points. During this time when there were periods when the silicomanganese alloy was not available and a mixture of ferromanganese and ferrosilicon had to be substituted. Thus, direct comparison of these two methods of deoxidation was afforded. Certain results taken from the heat records, and believed to be typical, are given in Table 33 following. Obviously, figures showing the variation in heat composition and the average manganese recovery of several years' practice can not be given in this table. The records themselves clearly show greater uniformity for the silicomanganese heats.

TABLE 33.—*Comparative results obtained with silicomanganese and with a mixture of ferromanganese and ferrosilicon.*[a]

Heat.	FeSi.	FeMn.	SiMn.	Total charge.	C.	Mn.	Si.	Mn. added.	Mn. recovered.	
	Lb.	*Lb.*	*Lb.*	*Lb.*	*Per ct.*	*Per ct.*	*Per ct.*	*Lb.*	*Lb.*	*Per ct.*
A	160	300	....	31,160	0.26	0.56	0.294	240	174	72.5
B	215	400	....	40,765	.21	.57	.306	320	232	72.5
C	160	310	....	30,620	.32	.63	.312	248	193	77.8
D	160	300	....	30,910	.27	.70	.318	240	216	90.0
E	215	400	....	40,865	.24	.72	.312	320	294	91.8
F	....	40	470	41,010	.22	.58	.308	281	238	84.7
G	....	35	350	30,685	.26	.60	.302	214	184	86.0
H	....	40	470	40,960	.21	.64	.310	281	261	93.0
I	....	40	470	30,510	.21	.66	.303	281	267	95.0
J	....	40	420	36,760	.24	.68	.310	255	250	98.0

[a] Approximate compositions: Silicomanganese, Mn 53 per cent, Si 20 per cent; ferromanganese, Mn 80 per cent; ferrosilicon, Si 50 per cent. Residual manganese was neglected in calculating recoveries.

It can hardly be claimed that these figures, or the three years' records which they represent with reasonable accuracy, furnish a truly scientific basis for comparison of the two alternate practices, but they do show rather convincingly that the same results (Mn and Si contents of the finished steel), by using silicomanganese can be obtained with consistently smaller amounts of both manganese and silicon, as compared with the combination of ferromanganese and ferrosilicon. In addition there is the advantage of more uniform practice, which in itself would warrant smaller additions. The weights of the additions favor the silicomanganese; thus in heats A, C, and D, 460 pounds was added, as compared with 385 pounds for heat G, and in heats B and E, 615 pounds was added, as compared with 510 pounds in heats F, H, and I. The low carbon content of the silicomanganese may or may not be a material advantage, but is in favor of the single-alloy addition because the carbon need not be worked as low and there seems to be less danger of missing the desired carbon content.

ELECTRIC-FURNACE PRACTICE.

No information is available to the writer bearing on the use of manganese-silicon alloys in electric-furnace practice, but we may at least consider such practice on the basis of the known behavior of such alloys. Considering acid casting practice first, there seems to be no reasonable doubt that either silicomanganese or silicospiegel could be at once substituted for ferromanganese and ferrosilicon. Inasmuch as the usual aim is to make high-grade castings, the manganese-silicon alloys would appear to have the distinct advantage of making sounder and cleaner steel. Silicospiegel, aside from possessing the theoretical advantage of being diluted with iron, could be more readily prepared with the correct manganese-silicon ratio so as to eliminate the use of an additional alloy. In this practice the advantage of greater dilution need not carry with it the disadvantage of increased weight on account of the higher temperature of the electric furnace. The uncertainty as to the relative behavior of the manganese-silicon alloys as compared with that of the ferro-alloys, and the relative efficiencies of low-grade and of high-grade alloys, as well as the importance of this step in the manufacture of steel, suggest the advisability of conducting a definite research to settle such points. It would seem that there is no better place for such a research than in this particular industry.

In basic electric-furnace practice the manganese-silicon alloys, on the same grounds, could likewise be utilized, particularly as the attempt is always to produce sound and clean ingots. However, in this practice, ferrosilicon is used as a reducing agent along with coke, and hence the operator would probably not see any advantage in changing his practice in favor of the manganese-silicon alloys.

BASIC OPEN-HEARTH PRACTICE.

The amount of information available on the use of silicomanganese in basic open-hearth practice is meager, but it can be said that silicomanganese can be used, probably with as satisfactory results as with ferromanganese and ferrosilicon. Through the cooperation of one steel plant the writer was able to follow two shell-steel heats made with silicomanganese which was added to the ladle. The results of the second of these heats are given here to show what was done. To 11,100 pounds of molten pig iron in the ladle was tapped 122,340 pounds (estimated) of steel analyzing 0.09 per cent C, 0.12 per cent P, 0.15 per cent Mn, 0.033 per cent S, and 0.02 per cent Si. At the same time 1,000 pounds of silicomanganese (50 per cent Mn), 300 pounds of 70 per cent ferromanganese, 12 pounds of aluminum, and 50 pounds of coal were added to the ladle. The heat was in excellent condition and the ingots had smooth even tops and displayed no superficial action or evolution of gas. The final analysis was 0.40 per cent C, 0.028 per cent P, 0.58 per cent Mn, 0.041 per cent S, and 0.21 per cent Si. The recovery of manganese, assuming the entire loss to come from the alloy added, was 77.5 per cent; the recovery of silicon was 65.1 per cent. Only 24 pounds of carbon, or 5 per cent of the total was lost. In the first heat, which was thought to be more highly oxidized, the recovery of silicon was only 58 per cent while the recovery of manganese was only slightly less. These results indicate that silicon protects manganese in "oxidized" heats.

SELECTED BIBLIOGRAPHY ON MANGANESE DEPOSITS.

GENERAL.

Beyschlag, F., Krusch, P., and Vogt, J. H. L. Die Lagerstatten der nutzbaren Mineralien und Gesteine. 3 vols., 1910, pp. 850–869, 1099–1115.

Demoret, L. Le principaux gisements des minerais de manganese du monde. Ann. des Mines de Belgique, t. 10, 1905, pp. 809–901.

Fach, E. Le mineral de manganese. Paris. 1914, 44 pp.

UNITED STATES.

Harder, E. C. Manganese deposits of the United States. U. S. Geol. Survey Bull. 427, 1910, pp. 298.

PANAMA.

Chibias, E. J. Manganese deposits of the Department of Panama, Republic of Colombia. Trans. Am. Inst. Min. Eng., vol. 27, 1897, p. 63.

Williams, E. G. The manganese industry of the Department of Panama, Republic of Colombia. Trans. Am. Inst. Min. Eng., vol. 33, 1902, pp. 197–234.

SOUTH AMERICA.

BRAZIL.

Branner, J. C. The manganese deposits of Bahia and Minas Geraes, Brazil. Trans. Am. Inst. Min. Eng., vol. 29, 1899, p. 756.

Scott, H. K. The manganese ores of Brazil. Jour. Iron and Steel Inst., vol. 1, 1900, p. 179.

Derby, O. A. On the manganese deposits of the Queluz (Lafayette) district, Minas Geraes. Am. Jour. Sci., 4th ser., vol. 25, 1901, p. 18.

CHILE.

Harder, E. C. Manganese ores of Russia, India, Brazil, and Chile. Trans. Am. Inst. Min. Eng., vol. 56, 1916, pp. 31–68.

EUROPE.

AUSTRIA.

Naske, T. [Manganese ore in Austria]. Stahl und Eisen, Jahrg. 28, 1907, pp. 543–547.

Scott, H. K. Manganese ores of the Bukowina. Jour. Iron and Steel Inst., vol, 94, 1916, pp. 288–355.

GERMANY.

Kern, J. Zur Frage der Mangan-versorgung Deutschlands. Berg. Mitteilungen, Jahrg. 4, 1913, pp. 49–59.

Scheffer, .— [The importance of manganese and manganiferous ores in German industry]. Gluckauf, Jahrg. 49, 1913, pp. 2056, 2111, 2151.

RUSSIA.

Drake, Frank. The manganese industry of the Caucasus. Trans. Am. Inst. Min. Eng., vol. 28, 1898, p. 191.

SPAIN.

Hoyer, M. Contributions tothe knowledge of the manganese deposits of the Province of Huelva. Ztschr. prakt. Geol., vol. 19, 1911, pp. 407–438.

AFRICA.

WEST AFRICA.

Ford, S. H. Manganese in West Africa. Mining Mag., vol. 17, 1917, p. 271.

ASIA.

INDIA.

Fermor, L. L. The manganese deposits of India. India Geol. Survey Mem., vol. 37, pts. I, II, III, IV, 1909,

Carter, H. A. Manganese mining in British India. Min. and Sci. Press, vol. 103, 1911, pp. 834–835.

India Geological Survey Record, Quinquennial review of the mineral production of India. Vol. 46, 1915, pp. 135–139.

JAPAN.

Snodgrass, J. H. Manganese mining in Japan. Mining World, vol. 30, 1909, p. 790.

PUBLICATIONS ON METALLURGY.

A limited supply of the following publications of the Bureau of Mines has been printed and is available for free distribution until the edition is exhausted. Requests for all publications can not be granted, and to insure equitable distribution applicants are requested to limit their selection to publications that may be of especial interest to them. Requests for publications should be addressed to the Director, Bureau of Mines.

The Bureau of Mines issues a list showing all its publications available for free distribution as well as those obtainable only from the Superintendent of Documents, Government Printing Office, on payment of the price of printing. Interested persons should apply to the Director, Bureau of Mines, for a copy of the latest list.

PUBLICATIONS AVAILABLE FOR FREE DISTRIBUTION.

BULLETIN 67. Electric furnaces for making iron and steel, by D. A. Lyon and R. M. Keeney. 1914. 142 pp., 36 figs.

BULLETIN 73. Brass furnace practice in the United States, by H. W. Gillett. 1914. 298 pp., 2 pls., 23 figs.

BULLETIN 77. The electric furnace in metallurgical work, by D. A. Lyon, R. M. Keeney and J. R. Cullen. 1914. 216 pp., 56 figs.

BULLETIN 84. Metallurgical smoke, by C. H. Fulton. 1915. 94 pp., 6 pls., 15 figs.

BULLETIN 97. Sampling and analysis of flue gases, by Henry Kreisinger and F. K. Ovitz. 1915. 68 pp., 1 pl., 37 figs.

BULLETIN 100. Manufacture and uses of alloy steels, by H. D. Hibbard. 1915, 78 pp.

BULLETIN 110. Concentration experiments on the siliceous red hematites of the Birmingham district, by J. T. Singewald, jr. 1917. 91 pp., 1 pl., 47 figs.

BULLETIN 133. The wet thiogen process of recovering sulphur from sulphur dioxide in smelter gases, a critical study, by A. E. Wells. 1917. 66 pp., 2 pls., 3 figs.

BULLETIN 140. Occupational hazards and accident prevention at blast-furnace plants; based on records of accidents in blast furnaces in Pennsylvania in 1915, by F. H. Willcox. 1917. 155 pp., 16 pls.

BULLETIN 154. Lead and zinc mining and milling, by C. A. Wright. 1918. 134 pp., 17 pls., 13 figs.

BULLETIN 157. Innovations in the metallurgy of lead, by O. C. Ralston. 1918. 167 pp., 13 figs.

BULLETIN 168. Recovery of zinc from low-grade and complex ores, by D. A. Lyon and O. C. Ralston. 1919. 145 pp., 23 figs.

TECHNICAL PAPER 8. Methods of analyzing coal and coke, by F. M. Stanton and A. C. Fieldner. 1913. 42 pp., 12 figs.

TECHNICAL PAPER 50. Metallurgical coke, by A. W. Belden. 1913. 48 pp., 1 pl., 23 figs.

Technical Paper 76. Notes on the sampling and analysis of coal, by A. C. Fieldner. 1914. 59 pp., 6 figs.

Technical Paper 86. Ore-sampling conditions in the West, by T. R. Woodbridge. 1916. 96 pp., 5 pls., 17 figs.

Technical Paper 96. Fume and other losses in condensing quicksilver from furnace gases, by L. H. Duschak and C. N. Schuette. 1918. 29 pp., 4 figs.

Technical Paper 102. Health conservation at steel mills, by J. A. Watkins. 1916. 36 pp.

Technical Paper 106. Asphyxiation from blast-furnace gas, by F. H. Willcox. 1916. 79 pp., 8 pls., 11 figs.

Technical Paper 135. Bibliography of recent literature on flotation of ores, January to June, 1916, compiled by D. A. Lyon, O. C. Ralston, F. B. Laney, and R. S. Lewis. 1917. 20 pp.

Technical Paper 136. Safe practice at blast furnaces; a manual for foremen and men, by F. H. Willcox. 1916. 73 pp., 1 pl., 43 figs.

Technical Paper 143. The ores of copper, lead, gold, and silver, by C. H. Fulton. 1916. 41 pp.

Technical Paper 149. Answers to questions on the flotation of ores, by O. C. Ralston. 1917. 30 pp.

Technical Paper 152. The inflammability of aluminum dust, by Alan Leighton. 1918. 15 pp.

Technical Paper 153. Occurrence and mitigation of injurious dusts in steel works, by J. A. Watkins. 1917. 20 pp., 4 pls.

Technical Paper 156. Carbon monoxide poisoning in the steel industry, by J. A. Watkins. 1917. 19 pp., 1 fig.

Technical Paper 157. A method for measuring the viscosity of blast-furnace slag at high temperatures, by A. L. Feild. 1916. 29 pp., 1 pl., 7 figs.

Technical Paper 176. Bibliography of recent literature on flotation of ores, July 1 to December 31, 1916, by D. A. Lyon, O. C. Ralston, F. B. Laney, and R. S. Lewis. 1917. 27 pp.

Technical Paper 177. Preparation of ferro-uranium, by H. W. Gillett and E. L. Mack. 1917. 46 pp., 2 figs.

Technical Paper 182. Flotation of chalcopyrite in chalcopyrite-pyrrhotite ores of southern Oregon, by Will H. Coghill. 1918. 13 pp., 1 fig.

Technical Paper 187. Slag viscosity tables in blast-furnace work, by A. L. Feild and P. H. Royster. 1918. 38 pp., 1 fig.

Technical Paper 198. Sulphur dioxide method for determining copper minerals in partly oxidized ores, by C. E. van Barneveld and Edmund Leaver. 1918. 12 pp., 1 fig.

Technical Paper 200. Colloids and flotation, by F. G. Moses. 1918. 24 pp.

Technical Paper 201. Accidents at metallurgical works in the United States during the calendar year 1916, compiled by A. H. Fay. 1918. 18 pp.

Technical Paper 211. Approximate quantitative microscopy of pulverized ores, including the use of the camera lucida, by W. H. Goghill and J. P. Bonardi. 1919. 17 pp. 3 pls.

Technical Paper 225. The vapor pressure of lead chloride, by E. D. Eastman and L. H. Duschak. 1919. 16 pp., 2 pls., 2 figs.

Technical Paper 227. The determination of mercury, by C. M. Bouton and L. H. Duschak. 1920. 39 pp., 2 pls., 1 figs.

Technical Paper 230. The determination of molybdenum, by J. P. Bonardi and W. L. Barrett. 1920. — pp., — pls., — fig.

Technical Paper 236. The abatement of corrosion in central heating systems, by F. N. Speller. 1919. 12 pp., 2 figs.

PUBLICATIONS THAT MAY BE OBTAINED ONLY THROUGH THE SUPERINTENDENT OF DOCUMENTS.

BULLETIN 3. The coke industry of the United States as related to the foundry, by Richard Moldenke. 1910. 32 pp. 5 cents.

BULLETIN 7. Essential factors in the formation of producer gas, by J. K. Clement, L. H. Adams, and C. N. Haskins. 1911. 58 pp., 1 pl., 16 figs. 10 cents.

BULLETIN 12. Apparatus and methods for the sampling and analysis of furnace gases, by J. C. W. Frazer and E. H. Hoffman. 1911. 22 pp., 6 figs. 5 cents.

BULLETIN 47. Notes on mineral wastes, by C. L. Parsons. 1912. 44 pp. 5 cents.

BULLETIN 54. Foundry-cupola gases and temperatures, by A. W. Belden. 1913. 29 pp., 3 pls., 16 figs. 10 cents.

BULLETIN 63. Sampling coal deliveries and types of Government specificaitons for the purchase of coal, by G. S. Pope. 1913. 68 pp., 4 pls., 3 figs. 10 cents.

BULLETIN 64. The titaniferous iron ores in the United States; their composition and economic value, by J. T. Singewals, jr. 1913. 145 pp., 16 pls., 3 figs. 25 cents.

BULLETIN 70. A preliminary report on uranium, radium, and vanadium, by R. B. Moore and K. L. Kithil. 1914. 114 pp., 4 pls., 2 figs. 15 cents.

BULLETIN 81. The smelting of copper ores in the electric furnace, by D. A. Lyon and R. M. Keeney. 1915. 80 pp., 6 figs. 10 cents.

BULLETIN 85. Analyses of mine and car samples of coal collected in the fiscal years 1911 to 1913, by A. C. Fieldner, H. I. Smith, A. H. Fay, and Samuel Sanford. 1914. 444 pp., 2 figs. 45 cents.

BULLETIN 98. Report of the Selby Smelter Commission, by J. A. Holmes, E. C. Franklin, and R. A. Gould, with reports by associates on the commissioners' staff. 1915. 525 pp., 41 pls., 14 figs. $1.25.

BULLETIN 108. Melting aluminum chips, by H. W. Gillett and G. M. James. 1916. 88 pp. 10 cents.

BULLETIN 122. The principles and practice of sampling metallic metallurgical materials, with special reference to the sampling of copper bullion, by Edward Keller. 1916. 102 pp., 13 pls., 31 figs. 20 cents.

BULLETIN 130. Blast-furnace breakouts, explosions, and slips, and methods of prevention, by F. H. Willcox. 1917. 280 pp., 2 pls., 37 figs. 30 cents.

BULLETIN 150. Electro-deposition of gold and silver from cyanide solutions, by S. B. Christy. 1919. 171 pp., 8 pls., 41 figs. 25 cents.

BULLETIN 171. Tests of rocking electric brass furnace, by H. W. Gillett and A. E. Rhodes. 1918. 131 pp., 4 pls., 1 fig. 25 cents.

TECHNICAL PAPER 31. Apparatus for the exact analysis of flue gas, by G. A. Burrell and F. M. Seibert. 1913. 12 pp., 1 fig. 5 cents.

TECHNICAL PAPER 41. The mining and treatment of lead and zinc ores in the Joplin District, Mo., a preliminary report by C. A. Wright. 1913. 43 pp., 5 figs. 5 cents.

TECHNICAL PAPER 54. Errors in gas analysis due to the assumption that the molecular volumes of all gases are alike, by G. A. Burrell and F. M. Seibert. 1913. 16 pp., 1 fig. 5 cents.

TECHNICAL PAPER 60. The approximate melting points of some commercial copper alloys, by H. W. Gillett and A. B. Norton. 1913. 10 pp., 1 fig. 5 cents.

TECHNICAL PAPER 81. The vapor pressure of arsenic trioxide, by H. V. Welch and L. H. Duschak. 1915. 22 pp., 2 figs. 5 cents.

TECHNICAL PAPER 83. The buying and selling of ores and metallurgical products, by C. H. Fulton. 1915. 43 pp. 5 cents.

TECHNICAL PAPER 90. Metallurgical treatment of the low-grade and complex ores of Utah, a preliminary report, by D. A. Lyon, R. H. Bradford, S. S. Arentz, C. C. Ralston, and C. L. Larson. 1915. 40 pp. 5 cents.

TECHNICAL PAPER 95. Mining and milling of lead and zinc ores in the Wisconsin district, Wisconsin, by C. A. Wright. 1915. 39 pp., 2 pls., 5 figs. 5 cents.

TECHNICAL PAPER 189. Temperature-viscosity relations in the Ternary system CaO-Al_2O_3-SiO_2, by A. L. Feild and P. H. Royster. 1918. 36 pp., 1 pl., 16 figs. 5 cents.

INDEX.

A.

Page.

Alloys, *See* Ferromanganese; Manganese alloys; Silico-manganese; Silicospiegel; Spiegeleisen.
American Iron and Steel Institute, war price schedule of........................ 10
American Manganese Co., acknowledgement to.. 108
Analyses. *See* Blast furnace gas; Blast furnaces, charge for; Coke; Concentrates; Manganese alloys, domestic; Manganese ore; Pig iron; Rhodochrosite; Slag, smelting tests of.
Arkansas, *See* Batesville, Arkansas.
Aubrey, Ga., flow sheet of mill at............ 48

B.

B. & B. Trading Co., acknowledgement to... 108
Bardwell, E. S., work cited.................. 165
Basicity, of slag, ratio to manganese, in ferromanganese runs............... 129
  in spiegeleisen runs.............. 145
Batesville, Arkansas, flow sheet of mill at... 50
  mining and treatment costs in........... 97
Bethlehem Steel Co., acknowledgment to... 108
Bibliography, on manganese deposits........ 200
Blast furnaces, burdening of, factors controlling............................ 131
  change of run from iron to ferromanganese in................................ 120
  charge for, analysis of..................... 134
    manganese balance in................. 135
  coke consumption of..................... 148
  driving rate of, ratio to basicity of spiegel slag................................ 145
  ferromanganese runs, analyses of ore for.. 112
    average costs for...................... 134
    fuel requirements for 120, 148
    operating data for...................... 121
    results of, accuracy of................ 110
    slag charge for, amount of.......... 112–114
    slag from, analyses of............... 118, 141
    slag from, ratio of manganses to basicity of.............................. 129
    temperatures used in...... 112–114, 118, 119
    See also Carbon: Electric furnace; Slag.
  gas from, analyses of.................. 123, 141
  hearth temperatures in................. 119, 141
  low-ash coke in, need for.................. 148
  manganese losses in, causes of............ 30
    See also Furnace losses; Slag loss; Stack loss.
  rate of driving. *See* Driving rate.
  slag from, analysis of...................... 135
    See also Slag.
  spiegeleisen runs, fuel requirements for 143, 148
    operating data on.............. 116–117, 143
    ore for, analyses of................. 116–117
    stack loss in.......................... 146
Blast furnaces, spiegeleisen runs, temperatures used in............116–117, 118, 143
    See also Carbon; Electric furnace; Slag.
  thermal reactions, formulas showing.... 121
  used in tests, dimensions of............... 109
Braunite, available oxygen in................ 8
  manganese dioxide in..................... 8
Brazil, manganese ore costs in.............. 101, 106
Buffalo Union Furnace Co., acknowledgment to.................................. 108
Butte, Montana, mining and shipping costs at 94

C.

California, ore from. *See* Manganese ores; Concentrates; Concentration results.
Cain, J. R., work cited........................ 190
Carbon, blast furnace charge, amount needed for, determination of............... 135
  burned at tuyères, amount of............ 142
  excess of, definition of.................... 126
    ratio to manganese in slag............ 145
      curves showing................... 127
  in ferromanganese furnace, amount absorbed........................... 121
  in manganese alloys, per cent............ 6
  ratio to silicon in ferro-silicomanganese alloys, curves showing.............. 157
  spiegel furnace runs, average weight used in.................................. 143
Carbon fuel, in blast furnaces, relation to weight of slag......................... 125
    relation to blast temperature........ 125
  for spiegel furnace, formula for............ 144
Carbonate ores, mining and shipping costs... 94, 100, 102
  occurrence................................ 94
Carbon dioxide, in blast furnace gas, percentage of................................. 123
Carbon monoxide, in blast furnace gas, percentage of.............................. 123
Carnegie Steel Co., price schedule, for ore, 1892–1917.......................... 9
Christianson, Peter, on Jones process......... 71–91
Cleaning methods, dry, description of......... 46
    disadvantages of........................ 46
  factors controlling........................ 47
Coke, blast furnace consumption of........... 148
  for iron furnaces, analyses of.............. 134
  requirements of, for manganese furnaces, estimate of.............................. 149
    in pig-iron industry..................... 150
Colorado Fuel & Iron Co., acknowledgment to 108
Concentrates from California ores, analyses of.................................... 39, 40, 41
  economic factors affecting................. 30
  costs................................ 38–43, 96–98
Concentration, commercial factors affecting . 37
  dry, manganese ores, description of..... 54
  with California ore. *See* Manganese ore.

Page.

Concentration, processes, classification of 36
  See also Cleaning methods, Jones process; Washing plant.
Copper ores, sulphur dioxide leaching of 66
Cornell University, chemical department, acknowledgment to 151
Corrosion, in sulphur dioxide leaching, prevention of 65
Costs. *See* Brazil; Concentration; Cuba; Electric smelting; Freight rates; Mining costs; Sulphur dioxide leaching; Wages.
Crane, W. R., on preparation of ore 45–56
  on production costs of ferrograde ores 92–107
Crimora mill, Va., flow sheet of 48
Cromlish, A. L., patent cited 178
Cuba, mining costs in 103
Cullen, J. F., work cited 152, 155, 168, 182
Cuyuna ores, two-stage electric smelting of 179–181

D.

Deitz, L. S., jr., acknowledgment to 151
Denzler, E. O., acknowledgment to 151
Dephosphorization, in electric smelting, tests of 172–178
Donner Steel Co., acknowledgment to 108
Driers, for oils, definition of 22
  use of manganese salts as 22–23, 24
Driving rate, for blast furnace, definition of 127
  determination of 127
Dry cell, description of 15–17
  See also Leclanche dry cell; Gassner dry cell.
  manganese dioxide for, requirements for 18, 19
Dyeing of cotton fabrics, manganese salts for 23, 24

E.

Eaton, W. R., acknowledgment to 5
Electric furnace, for manganese ores, description of 152, 157
  standardization of 159
  manganese-silicon alloys in 198
  See Slag smelting tests.
Electric smelting, for manganese ores, development of 151
  high-silica ores. *See* Silica, manganese ores high in.
  manganese, alloys made in, analyses of 184
    reducers for, tests of 163
  two-stage process, tests with 172–178, 179–181
  cost data for 186
  furnace data on 148
  tests of, description of 159
    ores used in 153, 185
    recovery of manganese in 153
    results of 184–185

F.

Ferro Alloys Committee, war-price schedule of 10
  ruling on ore payments 11
Ferromanganese blast-furnace runs. *See* Blast furnace, ferromanganese runs; Electric furnace; Slag.
Ferromanganese, composition of 6
  domestic production, 1905–1918 14
  imports, 1905–1918 14
  ore for, specifications for 7
  use with ferrosilicon 196, 197

Page.

Ferrosilicon, use with ferromanganese in steel making 196, 197
Ferrosilicon-manganese alloys, ratio of carbon to silicon in, curves showing 157
Fermor, L. L., work cited 8
Fluxing tests, high-silica ores 160–162
Foley, F. B., acknowledgment to 5
Freight rates, ocean, prewar, 1918 105
  railroad 11, 99, 100
Fuel requirements, ferromanganese furnace 120
  spiegeleisen furnace 143
Furnace. *See* Blast furnace; Electric furnace.
Furnace losses, prevention of 149
  See also Slag, manganese loss in; Stack loss.

G.

Gas. *See* Blast furnace gas; Carbon monoxide; Carbon dioxide.
Gassner dry cell, description of 16
Georgia, flow sheet of mill in 49
  mining and treatment costs in 97
Gillett, H. W., on electric smelting 151–187
Glass making, use of manganese dioxide in 20, 21

H.

Harder, E. C., work cited 154, 179
Harden, J., work cited 155
Haulage costs in manganese-producing States 96
  See also Freight rates.
Hausmannite, available oxygen in 8
  manganese dioxide in 8
Hearth temperatures. *See* Blast-furnace tests.
Hewett, D. F., work cited 13
Hoyt, Samuel L., on open-hearth steel practice 187–199
Hunter, W. H., on Jones process 71–91

I.

Imports, manganese ores, 1903–1918, statistics 13
Impurities, in manganese ore, treatment of 32–35
Iron, in manganese ore, treatment of 33
  in manganese alloys, per cent 6
  in manganese minerals, effect of sulphur dioxide on 61
  recovery in Jones process of 74–77
    curves showing 78–83

J.

Jigs, for standard washing plant 52
John B. Guernsey Co., acknowledgment to 108
Jones process, aim of 71
  apparatus for, description of 72
  duration tests, results of, curves showing 80, 81
  manganese recovered by, curves showing 78–83
  reducing tests, apparatus for, description of 87, 88
    conclusions on 84, 91
    procedure for 72, 87
    results of 85, 88, 90
      curves showing 79, 82, 83
  temperature tests, results of, curves showing 78, 82
Julihn, C. F., acknowledgment to 5

K.

Keeney, R. M., work cited 152, 155, 168, 182

L. Page.
Lang, G., patent cited........ 178
Leaching. *See* Sulphur dioxide leaching.
Leclanche dry cell, description of........ 15
Limestone for blast furnace charge, requirements for........ 137
  for blast furnace charge, tests of, results of........ 137
Logs, for standard washing plant........ 52
Lonergan, J., work cited........ 168
Louderback, T. S., acknowledgment to........ 5
Lyon, D. A., work cited........ 152, 155, 168, 182

M.

McKinney, P. E., work cited........ 25, 26
Mack, E. L., acknowledgement to........ 151
Manganese, conservation of........ 149, 191
  function in manganese bronze of........ 25
  function in steel making of........ 6, 29, 189
  loss, in blast furnace, causes of........ 30, 149
    See also Slag loss; Stack loss.
  metallic production 1905–1918........ 14
  recovery of. *See* Electric smelting tests; Jones process; Steel, open-hearth production.
Manganese alloys, domestic, carbon content of........ 6, 156
    composition of........ 6
    from electric smelting, analyses of........ 184
    from slag reducing tests, analyses........ 167
    prices, 1910–1918........ 12
    production, 1905–1918........ 14
    use in open-hearth steel practice........ 191
    use in steel industry........ 29
  imports, 1905–1918, statistics........ 14
  See also Ferromanganese; Phosphorus standard; Silicospiegel; Silicomanganese; Spiegeleisen.
Manganese borate, as a drier, preparation........ 22
Manganese bronze, composition of........ 25
Manganese chloride, use in dying cotton cloth........ 23
Manganese deposits, bibliography on........ 200
  types of, classification of........ 35
  location and origin, United States........ 27
Manganese dioxide, as drier, preparation of........ 22
  decolorizer for glass........ 20
  per cent in ores........ 8
  sources........ 19
  See also Glass making; Dry cell.
Manganese linoleate, as drier, preparation........ 23
Manganese minerals, solubility of........ 61
Manganese ore, California, analysis of........ 39, 40, 41
    concentration, costs of........ 39–43
      results of........ 39, 40, 41
    mining of, cost of........ 39–43
  cleaning methods for. *See* Concentration washing plants.
  definition of........ 28
  domestic, payments on, Ferro Alloys Committee ruling on........ 11
    production costs........ 94–101
    production statistics, 1912–1918........ 13
    prices, 1910–1918........ 12
    price schedule for........ 9, 10
    royalties on, sliding scale for........ 96
    transportation rates on........ 11, 99, 100, 105
Manganese ore, domestic and foreign, costs of........ 106
  for electric smelting tests, analyses of........ 184
    See also Electric smelting.
  foreign, mining and handling costs........ 102–106
  for glass making, specifications for........ 21
  high in silica. *See* Silica, manganese ores high in.
  imports, 1903–1918, statistics........ 13
  low-grade, classes of........ 157
  price schedule for........ 9, 10, 105–106
  slag-forming constituents in........ 34
  specifications for........ 7
Manganese oxalate, as drier, preparation........ 23
Manganese oxide, reduction, in blast furnace, formulas showing........ 124
Manganese persulphate, preparation and use........ 24
Manganese resinate, as drier, preparation........ 23
Manganese-silicon alloys, use in electric furnace........ 198
  use in open-hearth steel practice........ 195
  See also Silicomanganese; Silicospiegel.
Manganese sulphate, preparation and uses of........ 24
Manganite, manganese dioxide in........ 8
Marshall, E. E., acknowledgement to........ 108
Miami Metals Co., acknowledgement to........ 108
Mining, cost, California ores, data on........ 39–43
  effect of methods and marketing on........ 31
  for domestic oxide ores........ 96–98
Moissan, H. H., work cited........ 123
Montana, *See* Philipsburg, Montana.

N.

New Jersey Zinc Co., acknowledgement to........ 108
Newton, Edmund, on concentration and use of low-grade ore........ 27–44
  work cited........ 153, 154, 168
Nodulizing of fine concentrates, practicability of........ 44

O.

Open-hearth steel practice. *See* Steel, open-hearth production.
Oxide ores, costs in handling........ 95–98, 100, 102
Oxidizers, manganese salts as........ 24
Oxidizing ores, war prices of........ 12
Oxygen, in manganese ores, per cent........ 8

P.

Palmer, W. S., acknowledgement to........ 5
Pardee, J. T., work cited........ 154, 157
Penalties. *See* Phosphorus; Silica.
Pettijohn, Earl, work cited........ 190
Phalen, W. C., on uses of manganese ores........ 15–26
Philipsburg, Mont., flow sheet of mill in........ 51
  low-grade ores, analyses of........ 158
  mining and treatment costs at........ 98
Phosphorus, excess, penalties for........ 7, 11
  in manganese ores, effect in electric smelting........ 168
    effect of sulphur dioxide on........ 61
  standard, in manganese alloys........ 9, 29, 34
  volatization tests, in electric smelting........ 170–172
Picking belts, for washing plants........ 51
Pig-iron furnace, operating data for........ 121

Page.

- Pig-iron industry, annual coke requirements, estimate of 150
- Pig-iron slag, analyses of 118, 141
- Potassium permanganate, preparation and uses of 24
- Price schedule, for domestic ores 9, 10
- Probert, F. H., acknowledgement to 5
- Psilomelane, manganese dioxide in 8
- Pulp, from sulphur dioxide leaching, treatment of 60
  - drum for, description of 60
- Pyrolusite, manganese dioxide in 8
- Pyrometer, for high-temperature tests, description of 87

R.

- Reducers, for electric smelting, tests of 163
  - for slag-smelting tests, amounts used 167
- Reducing agent. *See* Jones process, tests with varying quantities of reducing agents.
- Reduction tests, Jones process 72–90
- Rhodochrosite, analysis of 164
  - manganese dioxide in 8
  - electric smelting of, difficulties of 164
- Rodenhauser, W., work cited 152
- Royalties on manganese, sliding scale for 96
- Royster, P. H., on allocation of low-ash coke 148–150
  - on blast furnace practice 108–147
- Ryan, F. C., acknowledgement to 151

S.

- Schoenawa, J., work cited 152
- Screens, for standard washing plant 51
- Seaboard Steel & Manganese Corporation, acknowledgement to 108
- Silica, in manganese ore, treatment of 34, 154
  - manganese ores high in, electric smelting of, acid slag in, advantages of 162
      - effect of phosphorus on 168, 169
    - electric furnace tests with 178–181
    - fluxing tests of 160
    - smelting of, difficulties of 154
  - penalties for 7, 10
  - premiums on, Ferro Alloys Committee ruling on 10
  - ratio to bases in manganese slag. *See* Slag, manganese in.
  - standard of Ferro Alloys Committee 9
- Silicomanganese, composition of 6
  - in open-hearth steel practice, use of 195
  - possible use of 156
  - production of, Sweden 155
    - United States 155
- Silicon, in manganese alloys, per cent 6
  - ratio to carbon in ferro-silicomanganese alloys, curves showing 157
  - reduction in electric furnace 154
- Silico-spiegel, composition of 6
- Silver in manganese ore, recovery of 33
- Sintering plants, for fine concentrates, practicability of 44
- Slag, acid, in electric smelting, advantages of 162
  - blast furnace charges of, analyses of 112–114
    - basicity of 118

Page.

- Slag, charge for ferromanganese furnace runs, amounts of 112–114
  - charge for spiegeleisen furnace runs, amounts 116–117, 143
  - from electric smelting, analyses of 184
  - manganese in, percentage of 126
      - formula for calculating 130
    - ratio to carbon difference, curves showing 127
      - results of furnace runs 129
    - ratio to basicity, results of furnace runs 129
    - ratio to rate of driving, curves showing 128
      - results of furnace runs 129
  - manganese, loss in 132
  - smelting tests of, analyses of 167
    - electric furnace, results of 167
    - *See also* Electric smelting.
  - volume of, relation to manganese loss 30, 34
  - weight of, method of calculating 110
    - relation to carbon fuel 125
  - *See also* Electric smelting; Ferromanganese; Pig iron, spiegeleisen.
- Smelting. *See* Blast furnace; Electric furnace; Electric smelting.
- Southeastern Iron Corporation, acknowledgment to 108
- Sperry, E. F., work cited 2
- Spiegeleisen, composition of 6
  - domestic production, 1905–1918 14
  - furnace tests. *See* Blast furnace, spiegeleisen runs.
  - imports, 1905–1918 14
  - molten, in open-hearth steel making, advantages of 191
  - ore for, specifications for 7
  - smelting of, description of 139
- Spiegeleisen furnace gas, composition of 142
- Spiegeleisen slag, composition of 141
    - manganese in, percentage of 144
    - ratio to basicity of slag and excess carbon 145
    - ratio to rate of driving 145
- Stack loss, causes of 132
  - definition of 132
  - furnace test of, results of 133
  - spiegel furnace, effect on manganese, recovery of 146
- Steel, function of manganese in 6, 29, 189
  - high manganese, manufacture of 192
  - open-hearth production of, acid, discussion of 195
    - basic, discussion of 198
    - investigations of 187
    - manganese recovery in 194, 197
    - use of molten spiegel for 191
  - production statistics, 1905–1918 14
- Storey, O. W., work cited 18
- Sulphur, in manganese ore, treatment of 35
- Sulphur dioxide leaching, complex ores, investigations of 66
  - plant for, costs of 67
  - plant for, flow sheet of 64
  - practicability of 70

Page.
Sulphur dioxide leaching, tests of, description of.......... 63
ores for.......... 59, 60
results of.......... 59
two-stage, advantage of.......... 65
two-stage, description of.......... 63
See also Sulphur dioxide for leaching.
Sulphur dioxide, effect on impurities in manganese ores.......... 61
for leaching, strength.......... 63
preparation of.......... 65
temperature of.......... 63
time of contact required.......... 63
Swann, T., work cited.......... 151, 154, 156

T.

Tabling, ores suitable for.......... 54
Temperature, *see* Blast furnace temperature.
Thermal reactions, blast furnace, formulas showing.......... 121
Thompson, J. G., acknowledgment to.......... 151
Transportation costs, *see* Freight rates; Haulage.

U. Page.
University of Minnesota, cooperation with.......... 71, 108
Utah, mining and treatment costs in.......... 97

V.
Van Barneveld, C. E., on sulphur dioxide leaching.......... 57–70
Virginia, flow sheet of Crimora mill in.......... 48
Volatilization, of phosphorus, tests of.......... 170–172
Vom Baur, C. H., work cited.......... 152

W.
Wages, in manganese districts, table showing.......... 95
Washing plant, design of, factors controlling.......... 47
flow sheets of.......... 48–51
standard, description of.......... 51
Water supply, importance in washing plants.......... 54
Weld, C. M., on production costs of ferro-grade ores.......... 92–107
on uses and prices of manganese.......... 6–14
Wharton, Steel Co., acknowledgment to.......... 108
Willcox, F. H., work cited.......... 183
Williams, C. E., on electric smelting.......... 151–186

Z.
Zinc, in manganese ore, treatment of.......... 33
in sulphur dioxide solution, removal of.......... 66